EXPLOSIVES AND BLASTING PRACTICES IN MINES

Samir Kumar Das

EXPLOSIVES AND BLASTING PRACTICES IN MINES

(With Modern Stoping Practices in the Underground Metalliferous Mines.
Revised Enlarged Edition covering all the modern thoughts and discoveries.)

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by

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Dedicated

to

My Mother, Late Father

Rumi and Sisters

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FOREWORD

The Explosive and Blasting technology in mines has made tremendous strides during the last few decades. The Indian Explosives industry has kept pace with world development and it is now using second and third generation explosives like slurry and emulsion explosives. The vast stretches of Indian mining industry is, however, continuing with the age-old practices oblivious of the recent developments.

Under these circumstances, a modern text-book on Explosive and Blasting which outlines the developments in explosives and blasting technology by giving appropriate weightage, both to conventional practices and to modern developments, would be the need of the day. The book entitled *Explosives and Blasting Practices in Mines* by Dr. Samir Kumar Das, an outstanding academician fulfils that need. The book should be a **Must** for every mining engineering student and also for the students aspiring for Overman's and Manager's Certificate of Competency, both for coal and metal mines. Academicians and researchers should go through the book to appreciate the ground realities of the Indian mining situation.

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PREFACE

Though various alternatives have been tried to eliminate the use of explosive since in the history of mining, until today with shearing and ploughing of coal seams, the use of explosive and consumption of this vital energy element will continue to rise significantly both in underground and surface mining for forseeable future. Last two decades have witnessed remarkable increase in the use of high explosives, improvement in rock blasting environment and changing state-of-the-art technology in blasting practices in mines. Publications and scriptures dealing with these are rather rare and backliners. The debut of the present compendium on "EXPLOSIVES AND BLASTING PRACTICES IN MINES" by **Dr. Samir Kumar Das,** is hence, an appropriate approach in this direction. The layout of the book in six chapters encompassing all possible technical turnouts make it more exact and effective both for students and practising mining engineers capable to meet most admirably the basic need of the subject. The untiring effort and unparallel attempts of the author leading to this valuable publication is commendable which I am glad to welcome and recommend for the entire mining and allied community directly or indirectly involved in explosive and blasting technology linked to mining and excavation.

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PREFACE

An attempt has been made in this book entitled 'Explosives and Blasting Practices in Mines' to guide the students of undergraduate and postgraduate courses in mining engineering, AMIE and Diploma level courses in mining engineering offered at various universities and also to the candidates who want to appear in various qualifying examinations conducted by the DGMS, with an uptodate information regarding explosives and blasting practices in mines and also latest stopping practices in the metalliferous mines. The second edition of the book has been further revised and enlarged covering all the modern thoughts and discoveries in this field which will definitely highly enrich the subject. I am confident that this book will serve the major need of the engineering students, teachers and practising engineers in this subject.

I would like to express my sincere thanks to all who encouraged, assisted, drafted figures and typed the manuscripts, without their help publication of this book would not be possible. My thanks will also go to the publisher of the book who helped me with valuable suggestions and providing a few writing materials. I am also acknowledging and expressing my sincere thanks to all the authors of the valuable papers and various catalogues and publications of various company and organisations mentioned in the 'Selected References and Acknowledgement' due to which writing of this book was possible. My thanks will go to the explosives companies like IDL-Chemicals Ltd., Karnataka Explosives Ltd., Maharashtra Explosives Ltd., Indo Gulf Explosives Ltd., who helped me a lot by providing their various catalogues and publications.

I am also expressing my sincere thanks to Dr. S. K. Sarkar, Ex-Head, Longwall Reserch Group and Condition Monitoring Unit, CMRS, Dhanbad and Mr. H. N. Karmakar, Ex-Chief Mining Engineer and Head, Design and Computers, CMPDIL, RI-II, Dhanbad, for writing foreword and preface of the book respectively.

Lastly, I am deeply indebted to my mother and my sisters for their encourragement and help which made this task a great success.

Samir Kumar Das

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Chapter 1

Explosives, Blasting Accessories, Transportation and Magazine

Question 1 : What is an explosive? What is meant by combustion and detonation? What materials are present in an explosive besides the explosive substance?

Answer : An explosive is either a solid, liquid or mixture of compounds which when subjected to a combustion (flame), detonation (sudden shock) or heat, undergo themselves to a rapid instantaneous change into a large volume of gases at high pressure and temperature. The combustion or detonation process initiate the chemical reaction among the compounds in the explosives. The released chemical energy, is converted to heat and mechanical energy during blasting and the released high pressure breaks the rock. The explosives should be very safe and sufficiently insensitive for any condition of handling and storage, possess adequate strength, VDO and density, good water resistance property, fume characteristics and storage qualities, etc. Combustion is the process of a very high rate of burning. Detonation is the process of propagation of a violent shock wave through a cylindrical charge initiating an instantaneous and high degree of chemical reaction and producing sufficient quantity of energy for the propagation of shock waves. If detonation is not there then there will be deflagration in the burning. A deflagration moves very slowly to produce shock waves sufficient to fracture rock. Under sufficient detonation rock shatters whereas in weak deflagration process rock displaces. An explosive contains the following ingredients besides the explosive substances :

a) Oxidising agents - Sodium Nitrate, Ammonium Nitrate, etc.

b) Combustible substances - Charcoal, Woodmeal, Sulphur, etc.

c) Stabilizers - Calcium Carbonate, Magnesium Carbonate, etc.

d) Sensitisers - Metallic Powder (Aluminium powder), etc.

e) Flame Depressant - Sodium Chloride, etc.

Question 2 : Discuss briefly about the main ingredients of an explosive.

Answer : An explosive normally consists of three main ingredients. They are :

1. Oxidiser
2. Fuel
3. Sensitiser

Oxidiser

Oxidisers are the carriers which help in supply of oxygen for the explosive reaction. Explosives in the confines of borehole can not depend on outside sources (such as air) for oxygen and the ingredients have to be self sufficient. Commonly used oxidisers are Ammonium Nitrate, Barium Nitrate, Calcium Nitrate, Sodium Nitrate, Sodium Perchlorate, Ammonium Perchlorate, etc.

Fuel

Commonly used fuels include Starch, Sugar, Fuel Oil, Glycols, Sulphur, Woodmeal, Aluminium, Ferro Silicon, etc.

Sensitiser

Sensitisers are added to the explosive mixture to make the explosive cap sensitive or booster sensitive as the case may be. Sensitisers also help in propagation sensitivity, detonation stability, ability to perform under tough conditions such as cold climates, high hydrostatic heads, etc. Sensitisers are of two types :

a) Explosive sensitisers such as Nitro Glycerine (NG), etc.
b) Non-explosive sensitisers such as Aluminium, Methyl Amine Nitrate (MAN), Hexamine, Air bubbles, Micro balloons, etc.

Slurry explosives normally contain few of the ingredients mentioned above in various proportions. In addition, thickeners such as gums are added and held together in an aqueous solution and gelled with the help of binding agents which are also called as cross-linkers. Most slurries which do not contain explosive sensitisers contain air bubbles which act as 'HOT SPOTS'. The adiabatic compression of air bubbles or artificial void by the shock wave imparted by an initiator results production of a very high temperature and generation of many such hot spots in the explosive medium, sets up a reaction in the adjacent material

and detonation progresses through explosive medium.

Question 3 : What characteristics are most importent for the selection of explosives? What are the most prominent factors on which the blasting performance depends? Discccuss.

Answer : The following characteristics are very important for the selection of explosives :

- Physical characteristics : Charge diameter, water resistance, fume characteristics, temperature sensitivity, etc. This knowledge can be obtained by the specification supplied by the manufacturer.
- Detonation characteristics : VOD, density, energy, energy partition, etc. Energy is the basic criterion to know the detonation characteristics.
- Economic and logistic characteristics : Cost, availability, reliability, ease of transport and handling, technical support by the supplier etc. are important. These characteristics are to be assessed to optimize the overall blasting operations in terms of cost and performance.

The blasting performance of an explosive is the function of the objective of the blasting operation, explosive-rock interaction, partition of explosive energy and the degree of confinement.

The basic objectives of blasting depend on the nature of use and can be explained as follows :

(i) For production blasting : requirement is the good fragmentation, loose muckpile with standard sizes to suit heavy earth moving equipment and crushers, no back break, no toe, no fly rock, not much of vibration, no secondary blasting afterward, well muck profile, generation of less fine (in case of coal etc.).

(ii) For overburden cast blasting : requirement is the perfect throw of rock into the desired demineralized zone.

(iii) In case of controlled blasting : the main requirement is the perfectly creating a controlled fracture without causing much damage to the surrounding rock-mass.

It's quite appropriate to state that 21% to 23% of the explosive energy does the useful work (in the initiation of fracture in situ, breakage

and rock mass displacement), 2% to 3 % explosive energy is wasted for generation of fly rock and deformation of solid rock around the shotholes, and remaining explosive energy is wasted in the form of ground vibration and air blast.

The explosive rock interaction is a function of the detonation properties of the explosive, the physical, mechanical and structural properties of the surrounding rock-mass and the degree of confinement. The various phases of it are as follows :

(a) Shock phase : The rock breakage mechanisms that can be attributed to the shock energy of the explosives are crushing, radial fracturing, spalling and release of load. The sudden impact of the high pressure gases on the borehole wall transmits a shock wave tnto the surrounding rock. When the intensity of stress is more than the dynamic compressive strength of rock the out-going shock wave crushes the surrounding rock and also develops radial fracture due to tangential - tensile stresses. Spalling may occur when the compression stress wave of sufficient amplitude is reflected back by a free face or an open joint causing the formation of tensile and shear waves provided that the intensity of the tensile waves exceeds the dynamic tensile strength of the rock. These tensile and shear waves are also responsible for extending the pre-existing cracks.

(b) Gas phase : Just after the transmission of shock wave the high pressure gas inside the boreholes wedges the radial fractures and/or natural fractures, open and extend them. The explosion gases push the rock-mass forward, cause flexural rupture, and assist formation of muckpile. It may also cause shear fracturing of the rock-mass when adjoining rock is displaced at different times or at different rates. Secondary breakage of rock may take place when the rock particles collide in motion.

(c) Burden movement phase : Regardless of blasting environment with the use of same kind of explosive the burden velocity decreases with increasing burden and this decreasing rate depends on geo-mechanical properties of rock, type and characteristics of explosives. The formation of muck pile depends on the time and velocity of burden rock movement.

The blasting energy can be partitioned into different zones as mentioned in the pressure volume curve (Fig 1). The area under the curve represents the total energy released by the explosives. The zones shown in the diagram represent the expansion of the explosion gases during the various stages of rock breakage. Just after the detonation of the charge, the chemical reaction in the exeplosives causes the production of heavy amount of gases which expand into the borehole and transmit a shock wave into the rock-mass. The volume of the blastholes starts increasing with a corresponding decrease in the pressure of the explosion gases until it reaches a quasistatic equilibrium state (i.e. where the pressure of the explosion gases in the borehole cavity is equalised by the stress in the surrounding rock-man). The main properties of rock include compressive strength, tensile strength, shear strength, young's modulus, p-wave velocity, density, poisson's ratio, insitu block size, fracture toughness, etc.

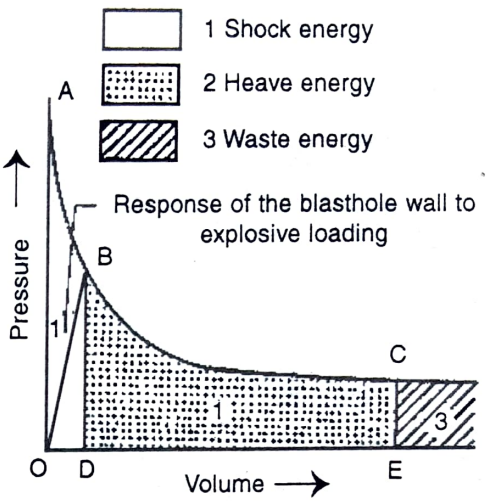


Fig. 1 Partition of Explosive energy released during blasting

Position A on the curve represents the initial explosion state, position B represents the equilibrium state and position C represents the points at which the gases escape into the atmosphere. The line OB represents the response of the blasthole wall to the explosive loading and is dependent on the stiffness of the rock. The energy delivered up to this point is termed shock energy (SE) and is represented by zone 1 (Area OABD).

Subsequent expansion of gases into the fracture network continues until they vent through a free face and escape into the atmosphere. Position D on Fig. ..... represents the volume of explosion gases at the equilibrium state and position E represents the volume of explosion gases at the time of venting into the atmosphere. The energy delivered up to this point is described as heave energy (HE) and is represented by zone 2 (Area DBCE). Further gas expansion after they vent into the atmosphere represents wasted energy and is represented by zone 3. The combined area represented by zones 1 and 2 is considered to be the effective energy released by the explosive in a given blasting situation.

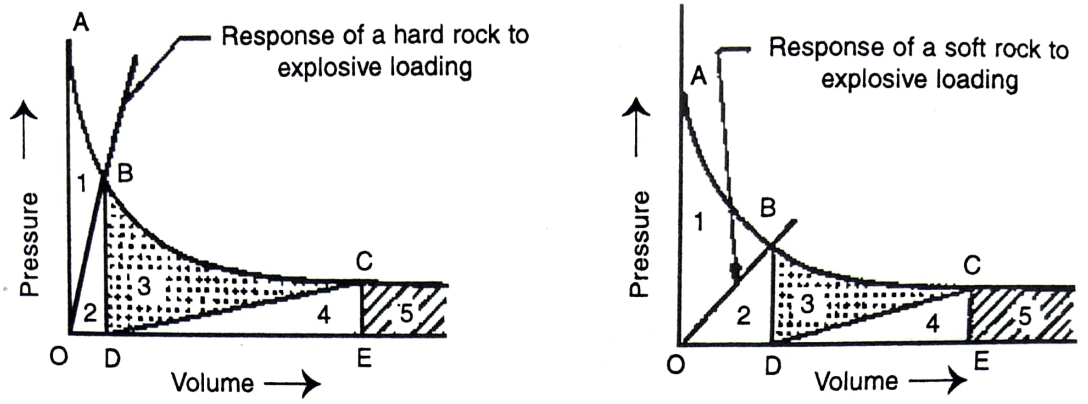


Fig. 2 & Fig. 3 Represents the partition of energy for different explosives in hard and soft rock repectively.

The pressure of the explosion gases depends on the degree of confinement i.e. by designing optimum burden and stemming. The lessly confined explosive charge allows the explosion gases to expand adiabatically and to vent into the atmosphere at high pressure causing lowering of heave energy, production of air blast, flyrock, ground vibration and noise. However a highly confined blast may not allow the movement of burden and the high pressure gases present inside the blasthole will cause radial and/or natural fracture and will damage the unwanted area of the rockmass. An optimal confined blast moves the burden rock at the right time and produces the desired muckpile with nominal damage. If the rate of gas venting is vary high then the production of larger size fragments and decrease in movement of blasted rock is observed. When stemming retention time is reduced below the initial burden movement time, significant increase in fragment size and decrease in rock movement will be the out come. However the optimum stemming height and retention capacity will result finer rock fragments and increased burden velocities. The burden rock velocity is inversely proportional to the stemming ejection velocity. The fig. 3(a) reprsents the effect of stemming efection of pressure time history of explosion gases.

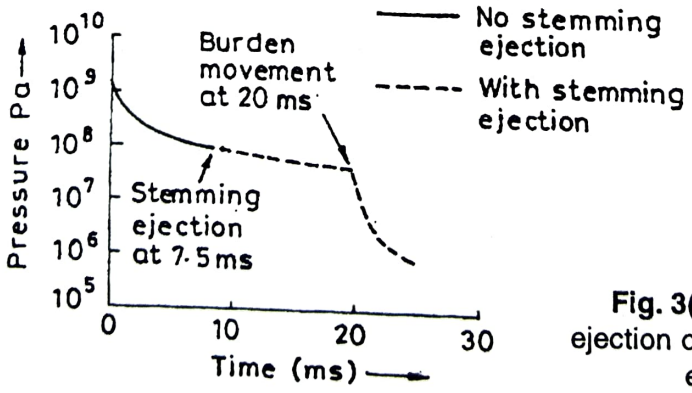


Fig. 3(a) Effect of stemming ejection of pressure-time history of explosion gasses

Question 4 : Classify the explosives and their accessories.

Answer : Under the explosives rules explosives and their accessories are classified as follows :

Class 1	Gun powders
Class 2	Nitrate Mixtures (like ANFO, Aquandyne, Energel, GN-1, Godyne, Permadyne, Permaflow, Powerflo, Powerite, Superdyne, Supergel, Toeblast, etc.)
Class 3	Nitro compounds
	Div : 1 Blasting Gelatine, Special Gelatine, OCG, Ajax-G, Viking-G, Soligex, Unisex-g, etc.
	Div : 2 Gun Cotton, PETN, TNT, etc.
Class 4	Chlorate mixtures
Class 5	Fulminate
Class 6	Ammunition
	Div :1 Safety fuse, Igniter cord, Connentors, Electric lighters, etc.
	Div :2 Cordtex detonating fuse, Plastic igniter cord, Fuse igniters, etc.
	Div :3 Detonators, Delay detonators, Relays, etc.
Class 7	Fire works
Class 8	Liquid Oxygen Explosives (LOX)

Question 5 : Write short notes on (a) Nitroglycerine (b) Ammonium Nitrate.

Answer :

Nitroglycerine ($C_3H_5(NO_3)_3$)

Nitroglycerine is the product of the reaction between nitric acid and glycerine compound. Its freezing point is 55°F and the specific gravity is 1.6. It is a type of fluid resembling oil and is not soluble in water. It is very dangerous to handle in pure form since it is very sensitive to shock and may explode and this sensitivity highly increases when it freezes. So to suit the industrial purposes some inert material should be mixed with it. Nitroglycerine based explosives are available either in the form of gelatinous, semi-gelatinous (10 to 30% NG), or powdery form, Higher percentage of Nitroglycerine content (more than 30%) in a Nitroglycerine based explosive will be in gelatinous form whereas the explosive with content less than 10% Nitroglycerine will be in the powder form.

Gelatinous nitroglycerine based explosives have better water resistant capacity. Nitroglycerine on explosion has a very good shattering effect and it is used as a base charge. On explosion, if Nitroglycerine reacts in perfect order, produces the following gases :

$$4C_3H_5(NO_3)_3 \Rightarrow 12CO_2 + 10H_2O + 6N_2 + O_2 + \text{Heat}$$

Ammonium Nitrate

Ammonium Nitrate (NH_4NO_3) is a very high explosive. It is also a very good oxidising and cooling agent and very safe to handle. To convert the NH_4NO_3 into an explosive charge it must be mixed with a sensitiser like Fuel oil, Nitroglycerine or Trinitrotoluene. It is very hygroscopis in nature. So in watery condition it must be kept under the waterproof covering. It has a low temperature of detonation and has less power compared to the Nitroglycerine. It is very cheap, safe to handle and give better fragmentation.

On perfect explosion Ammonium Nitrate produces the following gases :

$$2NH_4NO_3 \Rightarrow 2N_2 + 4H_2O + O_2 + \text{Heat}$$

Question 6 : Define Strength, Weight Strength, Water Resistance, Velocity of Detonation, Density, Sensitivity, Detonation Pressure, Column depth, Decking of an explosive.

Answer :

Strength

The strength of an explosive is the measure of the amount of energy produced by an explosive to do the useful work. The strength can be determined by the Weight strength, Bulk strength, Absolute weight strength and the Absolute bulk strength.

Weight Strength

It is the measure of the strength of any weight of explosive relative to the strength of the equal weight of the blasting gelatine. The blasting gelatine is used here as the standard explosive for comparison since it is the highly powerful explosive. It is determined by ballistic mortar test.

Water Resistance

Water resistance is simply the amount of time an explosive can be loaded in water and still detonates.

It is the capability of an explosive without losing any sensitivity and efficiency to withstand exposure to water. Gelatinous types of explosives have better water resistance characteristics. But gelatinized

high explosives like emulsion explosives do not have very high water resistance properties. ANFO has a very poor water resistant properties.

Velocity of Detonation (VOD)

Speed at which the detonation wave propagates through a cylindrical charge is called the velocity of detonation. Shock energy of detonation which does the most of the work under the unconfined state, increases in accordance with the velocity. VOD in most of the high explosives ranges generally in between 2500 to 7500 m/s. Slower and faster VOD is required for obtaining the heaving and shattering effect respectively. With the use of larger high density charge diameter along with large primer under the good confinement in the blasthole, the VOD can be increased. VOD depends upon the factors like the diameter of the hole, degree of confinement, the density and constituents of the charge, etc.

Blasthole velocity of detonation measurement has been used to evaluate the consistency of performance of bulk emulsion and other slurry explosives, the effect of amount of primer in the total explosive column, toe condition, susceptibility of emulsion and slurry explosives to dynamic desensitizations, etc.

Density

Density is a deciding factor for the explosive regarding the concentration of energy of a shot. For blasting hard ground high density explosive is used since it concentrates the energy whereas for soft ground low density explosive is desirable since it distributes the energy all along the blast hole. It is the ratio of explosive density to the specific gravity of water. By relating the density with the diameter of the blast holes the amount of explosives consumed permeter of the blast hole length can be determined. The density of most of the explosives varies in between 0.75 to 1.6 gm/c.c.

For sinking into wet bore-hole higher density explosives are needed.

Sensitivity

Senditivity is the measure of the energy required for setting off an explosive material. It is the explosive's propagating ability. ANFO has low cap sensitivity.

Explosive should be insensitive during normal handling, to friction, shock, etc. But it should be sufficiently sensitive while it is detonated and should propagate VOD nicely from one cartridge to another and even through the gaps of the explosive column. So it is related to hazard, performance initiation, propagation, gaps, etc.

Detonation Pressure

Pressure produced by the detonation wave propagating through a cylindrical charge is called the detonation pressure. It generally varies between 5000 kg./cm^2 to 150,000 kg./cm^2. It depends upon VOD and the density of an explosive.

Column depth : It is the portion of hole that is filled with explosive or blasting agent.

Decking : Decking is the separation of column of explosives in the borehole. Decking is done to avoid loading of explosives in the softer strata formation/broken seam to reduce the probability of blown outs and to reduce the consumption of explosives.

Question 6 (A) : How VOD of an explosive is measured?

Answer : The basic purpose of VOD measurements are to assess the explosive performance to obtain the real initiation sequence and to identify the source of the ppv pulses in the composite acceleration records. The VOD of an explosive is measured in the following ways—

Fibre optic INSTANTEL's VOD probes are to be inserted into the explosive at predetermined distances from each other but on a radius passing through the centre of the origin of the blast. The fibre optic insertion guides are inserted into the charge and taped securely in case of curtridge type of explosives (Fig.4). These guides are threaded by the fibre optic cables. Taping of guides with the explosives body are not necessary in case of bulk loaded type explosives. When the explosive column continued to be detonated the blast emitted light struck each probe at a different time period. The VOD is the ratio in-between the distance of probes and the elapsed time and it is measured by the instrument directly.

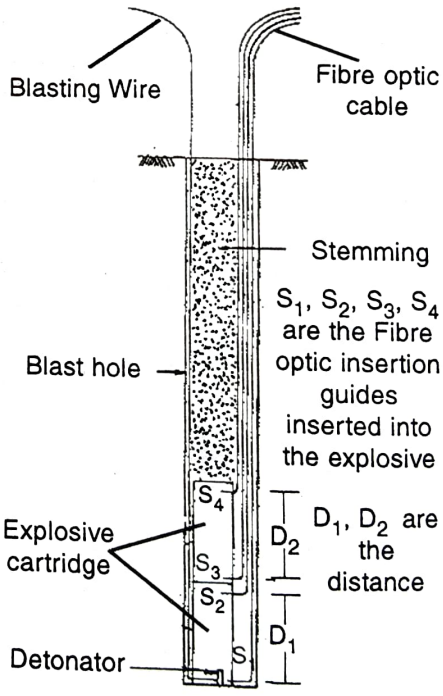


Fig. 4 Arrangement in the Blasthole for VOD Measurement

Question 7 : Discuss briefly about the (a) Low explosive (b) High explosive.

Answer :

Low Explosive

The effect of explosion energy and velocity of propagation of flame is low in this case. Explosives are generally fired by the ignition and the chemical reaction of the mechanical mixture of the substances of the explosive is done by the high rate of combustion and particles of the explosive body burn at their surfaces. Both the pressure and the temperature of the released gases are gradually built up.

Example : Gun powder.

High Explosive

The effect of explosion energy (shattering effect) and velocity of detonation of this type of explosive is very high. Explosives are fired by the detonation and the chemical reaction of the chemical ingredients of the explosive is done with high velocity and propagate instantaneously. Both the pressure and the temperature of the released gases are built up instantaneously. So within its own volumes the explosive compound converts into gases at a very high temperature and pressure

Example : Nitroglycerine based explosives, Slurry explosives, etc.

Question 8 : What types of blasting fumes are generally produced by an explosive? What is meant by Stability, Burden velocity, Loading density?

Answer :

Blasting Fumes

An explosive when initiated produces fumes of noxious and other gases like CO, CO_2, H_2O, SO_2, Nitrous fumes (NO_2), etc. depending on the presence of ingredients in the explosive. However, if the oxygen balance is optimum then a Nitroglycerine type of explosive will produce mainly CO_2, N_2, H_2O, etc. which are harmless, but if the balance of oxygen is either positive or negative then it will produce NO_2 and CO respectively (which are very poisonous) with lowering down the energy of the explosion. Nitroglycerine type of explosives when deflagrated and also blown out of them produce large amount of CO. ANFO with little amount of moisture content produces a heavy amount of reddish brown fumes which is an indication of concentration of highly toxic NO_2.

Stability

Chemical decomposition of any explosive is possible due to

prolonged storage particularly in heat and humid condition. The stability which is an important characteristic of an explosive will give an idea about upto what heat and humid condition and time of exposure, chemical composition of an explosive changes and quality of the explosive deteriorates. This deterioration of explosive will cause lowering bown of the energy of explosion and produces much amount of toxic blasting fumes, and even, it may be completely damaged. Nitroglycerine types of explosives are more stable compared to the other types of explosives.Ammonium Nitrate type of explosives when absorb moisture deteriorate faster. The deterioration sign is like softening of explosive cartridge, deposition of white incrustations over the cartridge, discharging of liquid form the cartridge. caking of Ammonium Nitrate type explosives, changing of colour (black) on the surface of the gelatinous cartridge, etc.

Burden velocity during movement - it indicates the suitability of selected blasthole burdens and it is the only method to assess the blasthole stemming performance. It will also help to optimize the cast blast design. It is assessed by high speed photography or video analysis.

Loading density : Peak blasthole pressure (^{P}bhole) depends on the density of explosive (^{r}exp) and the velocity of detonation (VOD) of the explosive.

$$^{P}\text{bhole} = 0.25\ ^{r}\text{exp} \times \text{VOD}^2$$

In smooth blasting the peak blasthole pressures are reduced to be only slightly more than the compressive strength of the rocks being blasted. This reduction is achieved by a reduction in the effective explosive density, either by diluting the explosive with an inert material or by decoupling the exolosive from the rock.

Question 9 : What are the main properties of a high explosive? How is VOD of an explosive measured?

Answer : The main properties are the following :

Strength, velocity of detonation, density, water resistance, fume quality, sensitivity, stability, resistance to freezing, etc.

Higher VDO is required for fragmentation of the stronger rock. The VOD can be measured by the following ways :

(1) By continuous measurement system.

(a) Time Domain Reflectometry system

(b) Shorted Location Indication by Frequency of Electrical Resonance.

(2) By Multi-point system.

In continuous measurement system a cable is inserted into the blasthole before charging by inverse initiation systems. As the explosive detonates the length of cable decreases continuously and the monitoring equipment attached with the cable produces a signal enabling the length of cable to be calculated at any instant. This technique is very much useful for having an idea of VOD over small intervals e.g. over the primer or at interfaces between different types of explosives.

In case of Multi-point system multicore cable is loaded into the blasthole at the time of charging the explosives with an electrode pairs located at known intervals up the explosive column. After blasting when detonation front reaches each electrode pair, surface equipment produces time pulses which can be recorded directly for waveform processing or which are used to trigger a timepiece that measures the time interval between the successive pulses with a very high degree of accuracy. By introducing an array of these electrode pairs over the full column length, the average VOD can usually be calculated with an accuracy of more than ± 1%.

Question 10 : Write few point about (a) Gun Powder (2) T.N.T. (c) Nitrocellulose.

Answer : Gun Powder (KNO_3 + S + C)

The substances which constitute a low explosive like gunpowder are as following :

Combustible mater	Approximate percentage by weight
Powdered Charcoal	15%
Sulphur	10%
Oxidising agents	
Potassium Nitrate	75%
(Sodium Nitrate	
may also be used;	
but it is more	
'hygroscopic')	

On explosion the following gases are produced :

$NaNO_3+C+S \Rightarrow Na_2CO_3+Na_2SO_4+Na_2S_3+CO_3+CO+N_2+Heat$

The gunpowder is very easy to manufacture, safe to handle, cheap, it has no adverse affect after blasting since it has heaving effect, and a large magazine is not necessary for it. However, it is easily damped by the water and moisture and it loses some of its explosive properties, produce flame which stands for a long duration, burning particles remain in the surroundings of the blasting site for a considerable time, produces many noxious gases after blasting, etc. It is not permitted to be used in the underground coal mines. It is fired by the safety fuse and a good quality gunpowder may develop a pressure of around 2000kg/cm^2.

T.N.T. ($C_7H_5N_3O_6$)

T.N.T. or trinitrotoluene is a very high explosive and it prodeces by the chemical reaction of nitric acid and toluene compound or benzene. It is used as an explosive base.

Nitrocellulose ($C_6H_7(NO_3)_3O_2$)

When nitric acid reacts with the cellulose compound it produces a high explosive named Nitrocellulose. It is used as an explosive base and gelatinizing agent.

Question 11 : Why nitroglycerine based explosives are very popular? Mention some of them. Give typical chemical composition and explosive properties of Ammonia Gelatine.

Answer : The nitroglycerine based explosives have high sensitivity to initiation, large explosion energy per unit volume (most of the

Table 1

Typical Chemical Composition and Explosive Properties of Ammonia Gelatine

Ingredient	Perecent	Property	Value
Ammonium Nitrate	4 - 20	Density, g/cc	1.3 - 1.6
Sodium Nitrate	55 -33	Detonation Velocity, m/s	2500 - 6000
Nitroglycerine	23-35	Water Resistance	Excellent
Nitrocellulose	0.3 - 0.7	Fume Characteristics	Moderate
Carbonaceous fuel	8.3 - 7.9		
Sulfur	7.2 - 0.5		
Antacid	0.7 - 0.8		

compositions have high densities, 1.25 to 1.45 gm/cc) and high detonation velocities and pressure, especially under confinement. Different types of NG based explosives are in use today viz : straight dynamite, ammonia dynamite, straight and ammonia gelatines and semi-gelatine. Typical chemical composition data and detonation properties are listed in the Table1.

Question 12 : What is LOX? Describe it. Where LOX can be efficiently used?

Answer : Like in any conventional explosive, liquid oxygen explosive (LOX) comprises of two main broad constituents viz. the combustible component and an oxidising agent. When the combustible component is saturated with liquid oxygen, each particle of the combustible substance is surrounded with an adequate amount of oxidising agent to ensure complete combustion. Under conditions of confinement such as in a borehole, when ignited or detonated, the reaction assumes explosive proportion resulting in the release of tremendous energy for blasting of rock/minerals. LOX has a VOD in the range of 4000 to more than 5000 m/s depending upon the liquid oxygen content.

The explosives and performance characteristics of liquid oxygen explosive depend to a very large extent on the choice of the combustible substances which must satisfy the following broad conditions :

- The explosives made from it should be very safe during the various operations of handling, transportation, storage and use.
- should have a high degree of inherent porosity so as to give a long and effective useful life.
- should easily mix with suitable additives to suit the varying strata conditions encountered in different mines,
- should be easily available.

IOL's Loxite explosive fulfils these requirements perfectly and these are now being extensively used in the various mining industries such as iron ore, coal, bauxite, magnesite, etc.

Question 13 : What is ANFO? Describe. Why cannot it be used in watery bore holes freely? Why is good quality prill important for producing ANFO? Enumerate the important properties for explosive grade prills. Tabulate the typical detonation properties of ANFO.

Describe briefly about the recent development of ANFO.

Answer :

ANFO

ANFO is a simple mixture of prilled Ammonium Nitrate (AN) and Fuel Oil (FO) nearly at the oxygen balanced ratio of 94/6 AN/FO. The 6% FO is important to raise the heat of explosion from 1.5 kJ/g for AN prills alone to 3.8 kJ/g for ANFO. Uniqueness of the present day ANFO lies in its major constituent—porous prilled AN. An important characteristic of these prills is that they have an available porosity of about 0.07 cc per gram and an available porosity of about twice this amount. The former is ideal for the absorption of the right quantity of FO necessary to optimise the strength of the explosive while the latter is ideal to give ANFO a low enough density that it detonates with optimum efficiency when properly initiated. It is thus obvious that a good quality prill is critical for producing ANFO. Both sensitivity and peroformance depend upon prill properties.

Properties important for explosive-grade prills are : low clay content, low moisture content, free-flow sizing oil absorbency, low particle density, optimum friability, noncaking tendency and stability towards temperature cycling. Typical detonation of ANFO are given in Table 2

Table 2
Detonation Properties of ANFO

Density g/cc	0.4 - 0.9	Detonation pressure, Gpa Theoretical	4.8
Detonation Velocity m/s	2000 to 4800	Gas volume, Its/kg	970
		Absolute weight strength, MJ/kg	3.8

ANFO does not detonate ideally and its performance properties strongly depend upon charge diameter and confinement. However, it provides good performance as an explosive at a reasonable cost. It is also a very good explosive for dry hole condition, safer to handle and has free flowing properties for loading. However, it has very less water resistant

property and for initiation high strength explosive (Primer) is required. A very good result is obtained if blastholes are fired as soon as they are loaded.

Reaction

On explosion ANFO produces the following gases :

$$3NH_4NO_3 + CH_2(x) \Rightarrow CO_2 + 7H_2O + 3N_2 + \text{Heat}$$

Ammonium Nitrate when reacts with aluminium produces the following gases :

$$3NH_4NO_3 + 2Al \Rightarrow 6H_2O + Al_2O_3 + 3N_2 + \text{Heat}$$

Previously AFNO used to be mixed and loaded into the holes crudely by hand but presently automatic blending and loading machines are developed which are widely being used worldwide. Fuel grade aluminium (particle size-20 mesh to +150 mesh) now a days adds with ANFO to increase the energy and density of the explosive. About 15% aluminium appears to be the economic limit from performance point of view. The major disadvantages of ANFO is that it is readily desensitized by water and therefore can not be used in watery bore hole. If it is to be used in watery borehole then it is to be packed in cartridge form with the help of polythene packet.

Now-a-days ANFO incorporates low density material such as polystyrene, sawdust, baggasse, etc. for using in weak overburden materials and saving cost. These low cost additives can be added in percentage as high as 50% by volume to ANFO without adversely affecting the fragmentation. For such mixtures the typical detonation velocities are around 2500 m/s at explosive densities in the range of 0.4 to 0.5 gm/c.c. in 200 mm dia. boreholes, compared to 3500 m/s for pure ANFO at a density of 0.829 gm/c.c.

Question 14 : What precautionary measures are to be taken during preparation and loading of the blast hole with ANFO?

Answer : The following precautionary measures are to be taken :

- Avoid hand mixing of ammonium nitrate and fuel oil. Use special automatic blending machine made by fire proof material to mix them in a remote and isolated place.
- The blending site should be free from any fire hazard.
- Remove prepared ANFO as soon as possible to the blasting site by a vehicle.

- Transport detonator, primer charge like OCG, special gelatine, etc. and detonating cord to the blasting site by a seperate vehicle.
- Load ANFO into the blasthole with an automatic loading machine.

Question 15 : Why slurry explosives and slurry blasting agents are developed? Describe them. What sensitizers are employed with this explosives? Give a typical chemical composition details and explosive properties of slurry explosives. Why slurry explosives are also called the watergel explosives? Why slurry explosives are slowly replacing the Nitroglycerine based explosives? How slurry explosive is loaded into the boreholes? What advantages may be offered by the pump truck system of loading?

Answer : Since the ANFO is hygroscopic in nature and has poor water resistant property, the development of AN based slurry explosives and slurry blasting agent (SBA) have come into the picture. These types of explosives incorporate beside oxidiser (ammonium nitrate, sodium nitrate, etc.), water, sensitizer, hydrophillic colloid such as guar gum which results in viscosity build-up of the matrix etc. This is followed by the addition of cross-linking agent which is usually a polyvalent metal iron and results in a cross linked gel network.

Water resistance arises from the fact that the cross-linking agent forms a network of bonds involving the polyvalent metal iron and the hydrated gum (polysaccharide) molecules, Proper density control is crucial for maximising the shock sensitivity of these explosives. This can be achieved in different ways, for example, mechanical aeration, chemical gassing agents and adding surfactants. Sensitizers employed include molecular explosives such as trinitrotoluene (TNT), monomethyl ammonium nitrate (MMAN) and finely divided metal powders such as flake aluminium. Saveral mechanisms have been indentified to be of importance in the case of initiation by high strength stimulus produced by blasting cap and these include shock/discontinuity, interactions, besides adiabatic compressive heating of air voids. Typical chemical composition details and explosive properties of slurry explosives are given in Table 3. The slurry explosives are also referred to as watergels since the matrix involves a gel network although suspended oxidizer/fuel particles are present.

Table 3
Chemical Composition and Properties of Slurry Explosives

Ingredient	Perecent	Property	Value
Ammonium Nitrate	30 - 50	Density, g/cc	1.05 - 1.25
Sodium Nitrate	5 - 15	Detonation Velocity, m/s	3800 - 4200
Water	10 - 25	Water Resistance	Good
Fuels	2 - 5	Fume Characteristics	Excellent
Flake Aluminium	2 - 5		
Monometryl Ammonium Nitrate	20 - 30		
Guar Gum	1 - 3		
Crosslinking Agent	0.05 - 0.15		

[Note : Depending upon the presence of ingredients, a density upto 1.55 gm/cc and a VOD of more than 5000 m/s is possible.]

Slurry explosives are amoug the safest of all explosives as they are not easily ignited and are almost completely insensitive to the types of shock, bullet, impact and friction stimuli that can initiate explosion of dynamites. In addition to the excellent safety characteristics of these explosives, they have good fume properties, water resistance, etc.

The slurry explosives (SE) offer greater safety during manufacture and usage, is leading to the gradual replacement of NG explosives by them world over inspite of their possessing slightly lower weight/volume strength. Another significant factor is that these explosives are also amenable to bulk mixing and loading procedures. Slurry automation is accomplished by the pump truck method. The ingredients such as oxidizer solution and fuel stored in sepearate tanks, are fed by auto-matically controlled pumps and auguers or by vibrators to a mixing funnel where they are picked up by a pump and forced rapidly through a long hose into the borehole while mixing and thickening are still in progress. Remarkable time-controlled thickening and gelling have led to the successful field implementation of the site-mixed slurry concept. The pump truck system permits precise control on both explosive composition and density, good borehole coupling and high loading rates (in the range 150-300 kg/minute) and thus minimize the blasting efficiency.

Quention 16 : What is meant by emulsions? Explain. What are the unique features of an emulsion explosive? What are the unique properties of emulsion explosives? What is the basic formation of emulsions? In which condition repumpable and straight emulsions are used? What are the doped emulsions?

Asnwer : An emulsion is a two-phased system in which a dispersed phase (oxidiser) is distributed in a continuous phase (Fuel). Here both the oxidiser or the dispersed phase and the fuel or the continuous phase are liquids as shown in Table 4. So in another way it can be expressed as, an emulsion is an intimate mixture of the liquids that does not dissolve in each other. The water levels are reduced with the use of super saturated salt solutions. The proximity of each unit of oxidser and fuel approaches in molecular proportions. The unique properties of the emulsion explosives are due to the minute size of the nitrate solution droplets and their tight compaction within continuous phase fuel as shown in Table 5.

Generally emulsions exist in two major categories, one the hot mix or straight emulsion, and the other in the repumpable emulsion. Basic formation of emulsions are :

Oxidisers	:	90% - 93%
Fuels	:	9% - 4%
Surfactant	:	1% - 3%

Table 4

Explosives	Oxidiser	Fuel	Sensitizer
ANFO	Solid nitrate (salts)	Liquid (diesel oil)	
Slurry	Solid/Liquid (salt solutions, nitrate salts)	Solid/Liquid (aluminium) sensitiser	Solid/liquid (TNT) (EGMN) (MAN)
Emulsion	Liquid (Salt Solution)	Liquid (Oil/Waxes)	Air bubbles or Physical voids

Table 5

Explosives	Oxidiser Solution Size	Form	VOD (km/s)
ANFO	2 mm	All solid	3.2
Slurry	0.2 mm	Solid/liquid	3.3
Emulsion	0.001 mm	Liquid	5.0 - 6.0

Various combinations of the other ingredients can be used to produce either the hot mixed emulsion or repumpable emulsion. Selection of the product depends on the end use temperature. If one is to use an emulsion at a temperature over 50°C, the hot mixed emulsion would be selected. If a lower usage temperature is required, a repumpable emulsion would be selected.

Emulstions are also frequently doped with varying percentages of prilled AN for higher energy yields. The product has a high degree of water resistance and is of booster sensitive.

Doped emulsions are formed by doping with varying percentages of prilled ammonium nitrate with emulsions for higher energy yields. The product is of booster sensitive and has a high degree of water resistance property.

Question 17 : Discuss Emulsion Explosives.

Answer : An emulsion explosive is a mixture of oxidiser and fuel both which are in liquid from. Normally, oil and water do not mix. But with the help of an emulsifying agent an intimate mix of oil water is possible and hence its name Emulsion. In this context, the oxidiser solution is the water and liquid fuel is the oil. By addition of emulsifiers, an intimate contact between oxidiser and fuel is obtained.

There are two types of emulsions :

1. Water-in-oil type.
2. Oil-in-water type.

In the water-in-oil type of emulsions, a discontinuous phase of oxidiser is dispersed in a continuous oil phase. One of the characteristic feature of emulsion explosives is that microfine droplets of the oxidiser is surrounded by a very thin layer of oil phase. As an illustration, water-in-oil emulsion can be compared to a Honey comb, where the Honey comb represents the oil phase and Honey the oxidiser phase.

Characteristic Features of Emulsions

1. As already explained, a very intimate mix/contact between the oxidiser and fual exists in Emulsion explosives.
2. Both oxidisier and fuel are in liquid form.
3. The particle size of the oxidiser droplets are few microns and the fuel layer surrounding each droplet is less than thousandth of a millimetre thick.

Properties of Emulsion Explosives

1. *High VOD :* In view of the very fine particle size and intimate mix of oxidiser and fuel, the reaction (detonation) through the explosive medium is very high. Emulsion explosives have a velocity of detonation between 4500-5500 m/sec. This generates more shattering power.
2. *Excellent water resistance :* Since every droplet of oxidiser is covered by a layer of fuel (oil/wax), the water resistance of emulsion explosives is excellent and comparatively superior to dynamites and slurries.
3. *Safety :* Emulsion explosives are exceptionally safe during manufacture, transport and usage. They do not contain any explosive ingredient. They are also comparatively insensitive to accidental initiation by friction, stimuli and fire.

In emulsion explosives also, the commonly used oxidisers are Ammonium Nitrate, Barium Nitrate, Sodium Nitrate, Calcium Nitrate etc. Fuel oil, mineral oils, waxes are used as fuel. In certain types of emulsions, such as small diameter cartridges, waxes also help in giving a 'Body' to the explosive material. Various grades of aluminium in different proportions are added to increase the energy content of explosives. Sensitivity is enhanced by the addition of micro balloons (Hollow glass spheres), or chemical gassing agent.

Satellite Plant

The basic emulsion matrix is manufactured in the satellite plant. This is a non-explosive matrix. The manufacturing process is highly controlled with sophisticated instrumentation and automation. The Emulsion matrix is loaded on to the pump truck only after strict quality control release tests are conducted and all release norms are satisfied.

Pump Truck

The Pump truck is desired to produce

i) EMULKING : Straight Emulsion explosive

ii) SUPERMIX : Emulsion blended AN-FO (Heavy ANFO)

The pump truck consists of two large bins one each for Emulsion matrix and prilled Ammonium Nitrate. The capacity of each bin is roughly 4-8 tonnes. Seperate tanks are also provided for Diesel oil, water, bulking/gassing agents. All the systems are hydraulically operated.

Straight emulsion is charged through a hose pipe, mounted on hose reel. Boreholes can be bottom loaded where boreholes are very watery or top loaded where boreholes are dry or where water column is not much in the boreholes. The high density of the emulsion ensures satisfactory sinking and displaces water.

The pump truck also has the ability to deliver Emulsion/AN-FO blends. The emulsion percentage varying between 30-70%. This explosive is delivered through a system of hydraulically driven augers and holes are top-loaded by placing the discharge end of the auger directly above the boreholes.

The pump truck is also provided with Load cells or RPM based digital meters which accurately record the weight of the explosive delivered in a borehole. Explosive quantities can be pre-set and the micro processor control accurately stops discharge after preset quantity is delivered into the boreholes. The system highlights the following main merits :

Safe - only non-explosive emulsion matrix is carried by the pump truck.

Assured quality - The basic emulsion matrix is manufactured in the satellite plant. A series of stringent quality control/release tests are conducted and then only transferred to the pump truck for usage in the field.

Reliable - The operations on the pump truck are very simple, minimal and no sophisticated mixing of ingredients is carried out. The non-explosive emulsion matrix is only sensitised on the truck to make it an explosive in the case of straight emulsions and blended with ANFO in the case of Heavy ANFO.

Hence the probability of a defective product going down the borehole is very remote compared to other systems where the entire process of the manufacture of the explosive is made on the pump truck.

Delivery rates of 200-300 kg/minute can be achieved. Load cells/ digital meters indicate accurate amount of explosive being loaded into the boreholes.

The emulsion blended ANFO is a very unique concept which utilises the distinct advantages of the individual explosive types to bring about an explosive having high bulk strength as compared to straight ANFO. Straight emulsion explosive also has a high bulk strength.

Emulsion matrix can be carried in support tankers of 10-12 tonnes capacity (or larger) which is transferred to the pump truck at site thus saving time. There is no necessity of the pump truck returning to the Satellite Plant for a refill. Similarly, Prilled Ammonium Nitrate can also be carried in an ordinary truck and loaded into the bin as and when necessry. Simple mechanisation of this operation is also possible, which would ensure uninterrupted charging.

Pump truck/ support tanker combination helps in achieving high loading rate at the lowest possible times. Movement of the pump truck is minimal thus ensuring longevity and less proneness for mechanical disturbances during travel ensuring trouble free performance and accurate feeds.

Table - 6 : shows some product specifications of IDL Chemicals Ltd.

Table 6

Product Specification of IDL Chemicals Ltd.

Product	Density (g/cc)	VOD (m/sec)	Relative bulk strength (ANFO=100)	Water resistance
EMULKING-100	1.2-1.25	4500-5000	115	Excellent
EMULKING-200	1.2-1.25	4500-5000	140	Excellent
SUPERMIX-100	1.10-1.18	3000-3500	140	Average
SUPERMIX-200	1.25-1.30	4000-4500	140	Excellent

Supermix 100 is excellent for dry boreholes. It also works satisfactorily under moist conditions and boreholes containing 0.5-1.0 m of water. Supermix 200 and Emulking series have excellent water resistance and sleeping ability. They can be used in watery boreholes.

For reliable initiation and maximum release of energy, it is recommended that 500 gms Pentolite Boosters are used for boosting Emulking and Supermix explosives in the boreholes.

Question 18 : What is meant by Heavy ANFO? Show a relation between absolute bulk strength vs percentage emulsion in the blend. How ANFO differs form Heavy ANFO?

Answer : The blend of base emulsion with a density around 1.35 gm/cc (it is also booster insensitive until voids are added to provide a sensitizing mechanism) and ANFO which creates the voids is known as

Heavy ANFO. The production of Heavy ANFO is done by mixing ANFO and emulsion, which are stored in separate tanks on a bulk truck.

The emulsion occupies the interstices between prills of ANFO within the borehole. The blends can be used efficiently m the larger volume of blastholes, yield higher brissance, and VOD, fit in any site specific condition, yield better fragmentation and enhance movement of rock to suit blast casting.

The two constituents can be mixed in varying ratios depending upon sensitivity, energy, water resistance requirements and economics. Typical ratios being, Emulsion : ANFO 30:70; 40:60 and 25:75. While relative bulk strength and water resistance increase with increasing emulsion content but sensitivity decreases. Energy and sensitivity peak at a ratio of 60:40 and a density around 1.3 g/cc. Fig. 5 is a plot of absolute bulk strength (ABS) versus percentage emulsion in the blend. This figure also depicts the variation of the relative expansion energy bulk strength (REEBS) with emulsion content. The latter quantity is derived from expansion energy which refers to that portion of the explosive energy that does real work, i.e., when the explosive product gases are expanding from the detonation state to a pressure of 100 MPa. Clearly, ANFO and the corresponding aluminised compositions possess less effective energy compared to Heavy ANFO, although the former has higher explosion energies. Heavy ANFO has gained wide acceptance in the mining industry.

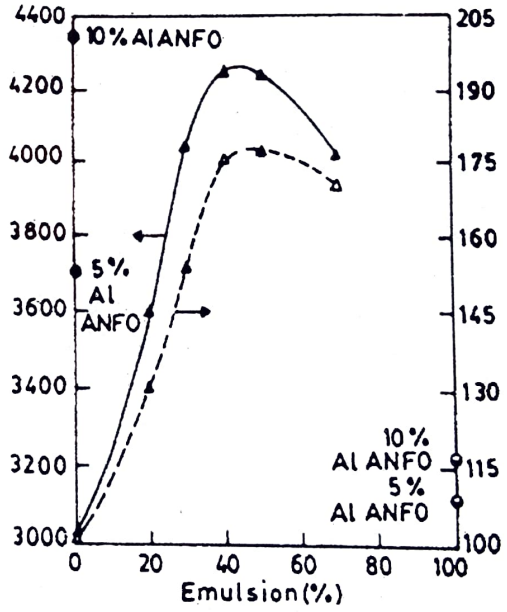


Fig. 5 : Plot of absolute bulk strength, ABS (J/cc) and relative expansion energy bulk strength REEBS (ANFO=100) as a function of emulsion content in Heavy ANFO

Question 19 : Give approximate value of shock energy and bubble energy of (a) Nitroglycerine based explosive (b) Aluminised watergel (c) MMAN based watergel.

Answer : The relative values of the three explosives are given in the Table 7.

Table 7

Explosive MJ/kg	Shock energy MJ/kg	Bubble energy MJ/kg	Total energy MJ/kg
NG based explosive	1.74	1.62	3.36
Aluminised Watergel	1.78	1.77	3.55
MMAN based watergel	1.81	1.57	3.38

Question 20 : Briefly show composion between ANFO and Heavy ANFO.

Answer : A comparision between ANFO and Heavy ANFO is given in the Table-8.

Table 8

Characteristics	ANFO	Heavy ANFO	Comments
Density (gm/c.c.) VOD (m/sec)	0.75-0.85 3000	1.05-1.25 3500-4500	
Relative weight strength (ANFO-100)	1.0	1.2	
Detonation stability	Fair	Good	Hence Heavy ANFO is very good for fractured strata.
Sensitivity	Booster sensitive. 20-30% Primer made of higher strength explosive is required. Cast booster can not be used	Booster sensitive. 20-25% Primer made of higher strength explosive is required. Cast booster can be used (0.2% of the total weight of explosives)	
Coupling in the borehole	Very good	Excellent	Better fragmentation is achieved in Heavy-ANFO
Explosive Energy (K Cal/kg)	900-920	1100-1150	
Safety during handling	Safe	Highly safe	Satic electricity may cause an accident.
Environmental friendly	Some toxic fumes are liberated	Very negligible toxic fumes are liberated.	
Relative Cost related to cost of production of minerals	Higher than Heavy-ANFO	Lower than Bulk ANFO	

Question 21 : What basic ingredients may be present in an emulsion explosive? Give a few properties of bulk emulsion product.

Answer : The following basic ingredients may be present in an emulsion explosive :

Ammonium Nitrate	60 to 70%
Calcium Nitrate/Sodium Nitrate	0 to 20%
Fuel Oil	2 to 6%
Aluminium	1 to 3%
TNT and water	varies

A few properties of bulk emulsion products are given in the table-9 on next page.

Question 22 : What is the bulk system of explosive? Explain what advantages it offers? How are the explosives amenable to bulk loading?

Answer : Bulk systems are essentially those where explosives are delivered directly into the borehole through mechanised and mobile delivery systems. Charging rates are fairly high and capacities of a single delivery unit range from 7 to 25t per load. In view of these features bulk loading systems are ideally suited to large explosive consumers with consumption levels of 2000 tpa or more although lower consumption levels may also be serviced at a higher cost. Properties of bulk emulsion products are given in the table -9.

If the density of the bulk explosives goes beyond 1.3 gm/cc the detonation may not be reliable and result in poor VOD, incomplete detonation and poor energy release. Products with low density are not selected since they have low bulk strength and also cause trouble in water filled hole. A product with a density ranging from 1 gm/cc to 1.25 gm/cc is ideal which will offer best detonation characteristics and energy release.

With ideal consumption levels bulk explosives offer the following advantages :

Feature	**Advantage with bulk**
• Safety :	Non-explosive ingredients are stored at all stages
• Inventory :	No inventory needs to be maintained and hence no investments in large magazines are necessory
• Explosive vans :	Large number of vans are not required
• Manpower :	Manpower savings are obtained with less deployment of van drivers/helper, blasting crew and magazine staff

Table 9

Properties of Bulk Emulstion Products

Type	Density (gm/cc)	VOD (m/s)	Water Resistance	Relative Weight Strength (ANFO-100)	Relative Bulk Strength (ANFO-100)	Example
Bulk ANFO	0.8	4700	Nil	100	100	-
Bulk slurry Bulk watergel	0.8-1.2	4500-5000	Excellent	80-100	110-135	SUPERBLAST, INDOGEL-634
Bulk emulsion/ Doped emulsion	1.15-1.25	5000-5500	Execellent	80-100 100-120	110-135 135-170	EMULKING, INDOGEL-1206 -
Heavy ANFO	1.00-1.30	4500-5000	Moderate	100-120	135-170	SUPERMIX

- Speed of operation : Swift charging rate of around 250-300 kg/minute
- Explosive product : Tailor made product with respect to density and energy can be delivered down the hole
- Blasting efficiency : Full borehole coupling 'can be achieved enabling expanded burden/spacing parameters
- Other features : No explosive pilferage
- Supply source and Magazine : Nearly Supply source and smaller magazine is required which will reduce the transportation cost, faster transport of explosives and require cheaper magazine.
- Operational satisfaction and bigger blast : Because of clean, less complicated methodology, safer technique, easy to handle and operate, blaster always becomes very happy with the system. It also offers to blast a huge number of blasthole at a time which definitely reduces the frequency of blast per weak and enhances higher output/blast, production time, production and productivity.
- Use of larger diameter blastholes : With this system large diameter blasthole can efficiently be used which will decrease drilling cost per tonne of output.
- Better fragmentation : With best blast design better fragmentation will be obtained. This will enhance the production and productivity of dumpers, shovels/draglines/pay loaders, cruchers, etc. with lower power cost, higher utilization of machine available time, higher bucket fill factor of shovel/dragline, less number of boulder generation and higher tonnage of material transport per trip by a truck/dumper.
- Environment friendly : It is environment friendly as very low quantity toxic fumes are produced.

Apart from the above positive features, bulk plants require fixed investments and with higher offtake levels, the cost get distributed over a wider volume enabling explosive manufacturers to offer lower prices. Hence cost savings are even greater at larger offtake levels.

Explosives amenable to bulk loading are ANFO, slurries, emulsions/doped emulsions and emulsified Heavy ANFO.

Question 23 : Give specification of some of the cartridge explosives used in the surface mines.

Answer : The specifications of some of the cartridge explosives which are used in the surface mines are given in Table 10.

Table 10

S. No.	Manufacturer	Type	Detonation Velocity (m/sec)	Density (g/cc)	Sensitivity	Classification	Cartridge diameter (mm)	Cartridge weight (kg)
1.	IDL Chemicals Ltd.	ENERGEL (slurry explosives)	3000-4000	1.15-1.20	Booster sensitive	Class - 2	83	2.78
							125	6.25
							175	8.33
							200 and above	12.5
		SUPERGEL (slurry explosive)	3000-5000	1.20-1.25	Insensitive to No. 6 and No. 8 detonator	- do -	75	2.08
							83	2.78
							125	6.25
							175	8.33
							200 and above	12.5
2.	Karnataka Explosives Ltd.	KELVAN-EXTRA	4200-4500	1.18-1.22	Insensitive to No. 6 detonator. Could be primed with KELVEX-800/ KELVEX-700/ KELVEX-P	- do -	83	2.78
							125	6.25
							200	12.50
		KELVEX-EXTRA	4000-4300	1.18-1.22		- do -	- do -	- do -
		KELVEX-500	4300-4500	1.20-1.23		- do -	- do -	-do -

Table 10 (continue)

SNo.	Manufacturer	Type	Detonation Velocity (m/sec)	Density (g/cc)	Sensitivity	Classification	Cartridge diameter (mm)	Cartridge weight (kg)
3.	Maharashtra Explosives Ltd.	MEXBLAST (watergel slurry explosive)	4000±200	1.25±0.03	Booster sensitive	Class - 2	125	6.25
							175	8.33
							200	12.50
		MEXCOL (watergel slurry explosive)	3800±200	1.15±0.03	- do -	- do -	83	2.78
							125	6.25
							175	8.33
							200	12.50
		MEXAN-II (watergel slurry explosive)	4000±300	1.18±0.03	Booster sensitive	- do -	83	2.78
							125	6.25
							175	8.33
							200	12.50
		MEXAN-I	3800±200	1.13±0.03	- do -	- do -	- do -	- do -
		MEXPRO in LDPE bags	3500±200	1.10±0.03	Nominal booster charge	-	83 & above	

Question 24 : Write short notes on

(a) Emulking (b) Superblast (c) Supermix

Answer :

Emulking

This type of pumpable bulk emulsion explosive comprises of an oxidizer which is dispersed in the fuel phase containing diesel/mineral oil and an emulsifier. Atomised aluminium powder is added with it for enhancing strengths. This type of explosive produces higher percentage of shock energy compared to slurries and Heavy-ANFO because of their higher VOD. This type of explosive offer higher detonation stability, longer sleeping time and excellent water resistant properties.

Superblast

It is a pumpable slurry explosive. This explosive besically consists of a saturated solution of Ammonium Nitrate or a mixture of oxidizer salt in which fuels and sensitizers are dispersed. The aqueous phase is thickened with gums and gelled with cross linking agents to keep the ingredients together in the form of a matrix, offer greater degree of cohesiveness and better water resistance. Atomised aluminium powder may be added with the ingredients for enhancing strength. The cross linking agent imparts higher product stability and water resistant properties. This type of explosive is very good in medium to hard rock conditions. This explosive may be sensitised on the pump truck to become booster-sensitive by the addition of chemical gassing or bulking agents.

Supermix

It is made by using prilled Ammonium Nitrate of very high quality. This prilled Ammonium Nitrate along with appropriate amount of Diesel oil is mixed with emulsion/slurry matrix on the pump truck in a special mixer which ensure uniform oil distribution in the prills. The mixing ratio of Emulsion/Slurry with ANFO should vary as per the strata conditions and water content in the blastholes. This product produce less shock energy and more heave energy and therefore this type of explosive is suitable for soft to medium hard rock conditions. This products has moderate water resistant properties.

Question 25 : Give an idea about the loading densities of explosives in kgs/metre of borehole.

Answer : Loading densities of explosives is given in Table 11.

Table 11

Borehole Dia in mm	LOADING DENSITIES IN KGS/METRE OF BOREHOLE Density of explosives in gm/cc.										
	0.90	**0.95**	**1.00**	**1.05**	**1.10**	**1.15**	**1.20**	**1.25**	**1.30**	**1.35**	**1.40**
32	0.72	0.76	0.80	0.84	0.88	0.92	0.96	1.00	1.04	1.08	1.12
40	1.13	1.19	1.26	1.33	1.39	1.45	1.50	1.56	1.62	1.68	1.75
50	1.76	1.86	1.96	2.05	2.15	2.25	2.35	2.45	2.54	2.64	2.74
75	3.96	4.18	4.41	4.63	4.85	5.07	5.29	5.51	5.73	5.95	6.17
100	7.06	7.45	7.85	8.24	8.63	9.02	9.42	9.81	10.20	10.60	11.00
125	11.04	11.65	12.27	12.88	13.50	14.11	14.75	15.34	15.95	16.65	17.17
150	16.86	17.76	18.65	19.55	20.59	21.49	22.38	23.28	24.32	25.22	26.11
175	21.64	22.84	24.05	25.55	26.45	27.56	28.86	30.06	31.26	32.46	32.67
200	28.27	29.85	31.42	32.99	34.56	36.13	37.70	39.27	40.84	42.42	43.99
250	44.18	46.63	49.09	51.54	54.00	56.45	58.91	61.36	63.82	66.22	68.72

Question 26 : Give specifications of some of the cartridge explosives used in underground non-coal mines.

Answer : The specifications of some of the cartridge explosives which are used in underground non-coal mines are given in the Table 12.

Table 12

SNo.	Manufacturer	Type	Detonation Velocity (m/sec)	Density (g/cc)	Sensitivity	Classification	Cartridge diameter (mm)	Cartridge weight (kg)
1.	IDL Chemicals Ltd.	Superdyne	3400-4000 (25 mm unconfined)	1.15-1.25	Sensitivity to No. 6 detonator & detonating fuse	Class - 2 Explosive	25	0.14
							40	0.50
							50	1.00
2.	Karnataka Explosives Ltd.	KELVEX-220	3800-4000	1.18-1.22	Sensitivity to No. 6 detonator & detonating cord	Class - 2 Explosive	25	0.14
							32	0.25
							50	1.00
	- do -	KELVEX-100	3200-3600	1.10-1.50	- do -	Class - 2 Explosive	- do -	- do -
3.	Maharashtra Explosives Ltd	MEXGEL	3600 ± 200	1.13 ± 0.03	No. 6 detonator at a critical temparature of 0^0C	Class - 2 Explosive	25	0.14
							32	0.20
							50	1.00
	- do -	MEXGEL-HI	4000 ± 200	1.15 ± 0.03	- do -	- do -	25	0.14
							32	0.20
							50	1.00
4.	Indo Gulf Explosives Limited	Detamite - 60 (Excellent water resistant, Weight strength -81% of B.G.)	4200 ± 200	1.25-1.28	No. 6 detonator		25	0.14
							32	0.20
							50	1.00
	- do -	Detamite - 80 (Excellent water resistant, Weight strength -88% of B.G.)	4300 ± 200	1.25-1.30	- do -	-	- do -	- do -

Question 27 : Give specifications of various types of permitted explosives used in the underground coal mines.

Answer : The specifications of various types of permitted explosives which are used in the underground coal mines are given in Table 13.

Table 13

S. No.	Manufacturer	Type	Detonation Velocity (33mm confined) ms	Density (g/cc)	Sensitivity	Classification	Cartridge diameter (mm)	Cartridge weight (kg)
1.	IDL Chemicals Ltd.	Godyne (Permitted P_1 explosive)	3050-3650	1.05-1.159	Sensitivity to No. 6 detonator	Class - 2 Explosive	32	150
	- do -	Permadyne (Permitted P_3 explosive)	3050-3650	1.05-1.159	Sensitivity to No. 6 detonator	Class - 2 Explosive	32	150
	- do -	Pentadyne (Permitted P_5 explosive) Fig.-6	3059-3650	1.05-1.59	Sensitivity to No. 6 detonator	Class - 2 Explosive	32	150 200
2.	Maharashtra Explosives Ltd.	MEXPERM-I (Permitted P_1 explosive)	3800 ± 200	1.13 ± 0.03	Sensitivity to No. 6 detonator	Class - 2 Explosive	32	200
	- do -	MEXPERM-III (Permittd P_3 explosive)	3600 ± 200	1.13 ± 0.03	Sensitivity to No. 6 detonator	Class - 2 Explosive	32	200
	- do -	MEXPERM-V (Permitted P_5 explosive)	3400 ± 200	1.10 ± 0.03	Sensitivity to No. 6 detonator	Class - 2 Explosive	32	200
3.	Indo Gulf Explosive Ltd.	Detacoal-I (For degree I gassy coal mine)	3200 ± 200	1.10 ± 0.03	Sensitivity to No. 6 detonator	-	32	250
	- do-	Detacoal-III (For all degree of gassiness)	- do -	- do -	- do -	-	32	250

Question 28 : What is meant by Ultra Safe Explosives, Sheathed Explosives and Equivalent to Sheathed Explosives?

Answer :

Ultra Safe Explosives

These explosives are disigned to give even a greater measure of safety to blast coal or coal measure strata in the underground coal mines. These types of explosives contain a very high percentage of cooling agent like finely grounded sodium chloride and nitrochalk.

Example : N.E. - 1235, Same specifications are given in the Table-14.

Table 14

Relative weight strength (percent B.G.)	28	Relative Bulk Strength (perecent B.G.)	21
Density (g/cc)	1.1	Velocity of detonation (m/s)	1700
Consistency	Powder		

Presently Maharashtra Explosives Limited of India is manufacturing three types of Ultra safe permitted explosives viz. MFXPERM-I, MEXPERM-III and MEXPERM-V (specification of which is given in the question No. 27 and in Table-15) for the use of

Table 15

Explosive	**Use**	**Maximum charge limit**	**Permitted group**
MEXPERM-I	Coal Seam of Degree-I gassiness	800	P_1
MEXPERM-III	-do-	1000	P_3
MEXPERM-V	Coal seam of Degree-I gassiness	1000	P5
	Degree-II gassiness	565	

underground coal mines. They are all aluminised cap-sensitive permitted explosives which are exceptionally safe. They have low detonation temperature and are insensitive to friction and heat as met with under normal conditions of usage. The explosives are so designed and balanced that they produce least amount of noxious gases eliminating headache/

nausea to operating personnel. They are waterproofed and their gel consistency offers improved borehole filling characteristics, thus ensuring optimum blasting results. Like all other permitted explosives they must be used with approved type of detonator.

Sheathed Explosives

These types of explosives come under the permitted explosives group P_2. A sheath of sodium bicarbonate ($NaHCO_3$) about 3 mm thick surrounds all along the length of the cartridge of a sheathed explosive which held in position the powdered sodium bicarbonate by a paper wrapper or in the form of felt attached to the cartridge by an adhesive. When the explosive is blasted the sodium bicarbonate decomposes ($2NaHCO3 \Rightarrow Na_2CO_3 + Co_2 + H_20$) and produces CO_2 and H_20 which blanket the flame and hot gases of the explosives all around it in the blasthole and also absrobs considerable heat from the hot gases produced by the explosives. Because of the sheath it protects against any breaks/ cracks in the shothole, they produce more amount of coal due to the cushioning effect of the sheath and also produce less amount of noxious gases. But the demerits in these types of explosives include requirement of larger diameter shothole, high cost and also sheath gets damaged easily due to the improper handling or tampering which causes incomplete detonation. These types of explosives are not produced in India.

Equivalent Sheathed Explosives

These types of explosives come under the permitted explosives group-P3. An inert material like mixture of ammonium chloride and sodium nitrate or ammonium nitrate and sodium chloride incorporates directly and uniformly throughout the composition of the explosives so that the explosion products render the explosives as safe as the sheathed explosives. The demerits associated with the sheathed explosives viz. the requirement of larger diameter shothole, damage of sheath due to improper handling and tampering, high cost, etc. are eliminated in these type of explosives.

Example :

Unisax-G by IEL Limited

Permadyne by IDL Chemicals Ltd.

For specifications consult question No. 27 and 159.

Question 29 : Give examples of some of the explosives used in the opencast mines showing their basic properties.

Answer :

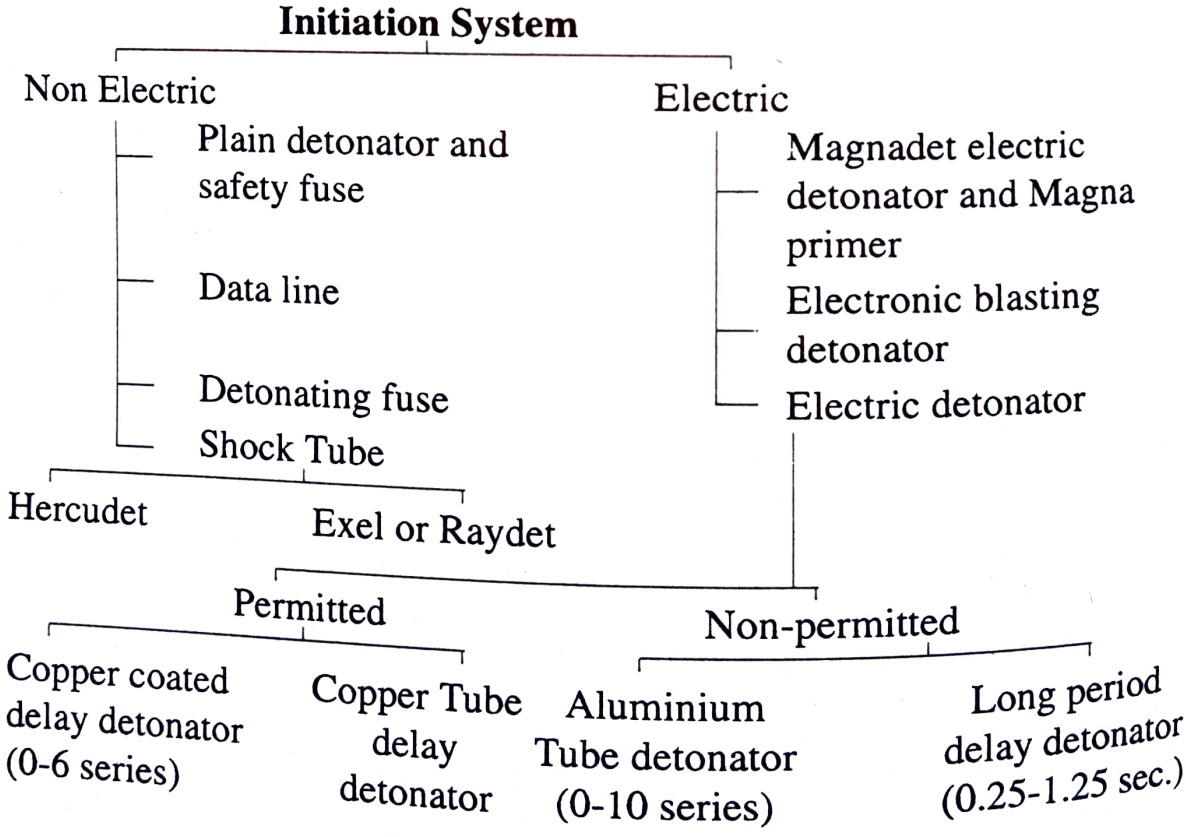
Example of few explosives manufactured by IEL :

Table 16

Product	Type	Density g/ml	VOD m/sec	Water proofness	Dia mm	Cartridge weight, kg	Usage
OCG	NG	1.4	6000	Excellent	75-200	2.5-12.5	As primer/ booster & toe charge
Pewerflo 1	Slurry	1.2	4000	Excellent	125-200	6.25-12.5	Column charge
GN/1	AN blasting Agent	0.95	3000	Good	83-200	2.78-12.5	Column charge
Powergel C	Emulsion	1.2	5000	Excellent	83-200	2.78-12.5	Primer/ booster
Powergel 2	Emulsion	1.2	4800	Excellent	83-200	2.78-12.5	Base cum Column charge
Powergel 1	Emulsion	1.2	4800	Excellent	83-200	2.78-12.5	Column charge

Question 29 (A) : Categorize the initiation system and give the characteristics of an electric delay detonator.

Answer :



Characteristics of an Electric Delay Detonator

(*Manufactured by the* IEL)

1. Electrical Resistance	:	Length of wire (m)	Resistance (ohm)
		1.8	2.5-4.1
		2.5	3.3-5.0
		3.0	3.6-5.7
2. Firing impulse	:	4-6 mw-sec./ohm	
3. Single firing current	:	0.5 mw-sec./ohm	
duration	:	50 ms.	
4. Series firing current	:	1.25 amp	
duration	:	4ms	
5. No firing current	:	0.3 amp	
duration	:	50 ms.	

Question 30 : What is meant by the detonators? For what purpose they are developed? Classify detonators. Briefly discuss about the various types of detonators.

Answer : A detonator is a small charge of special explosive enclosed in a small copper or aluminium tube. Due to the chemical reaction initiated either by the flame (safety fuse) or electric current, the special explosive explodes with high intensity and propagates detonation wave through the high explosives around it's external periphery and initiates the later.

Detonators are of two basic types:

- Plain detonators - for use with safety fuse. The spark of it initiates the detonator.
- Electric detonators - for blasting electrically.

Table 17 shows the various types of detonators which are manufactured by IEL.

Question 31 : Give some examples of conventional and non-conventional explosives.

Answer : Examples of conventional types of explosives are as following :

Opencast Gelignite, Toeblast, Aquadyne, Special Gelatine, Soligex, Unisex -G, Godyne, Permadyne, Pentadyne, MEXPERM-I, MEXPERM-III, MEXPERM-V, KELVEX-100, KELVEX-220, etc.

Examples of non-conventional types of explosives are as follows :

Table 17

Detonators (Manufactures by IEL)
(a) Plain Detonators : meant for use with safety fuse.
(b) Electric Detonators Manufactured in No. 6 and No. 8 strengths

Type	Dealy Nos.	Nominal Delay Interval	Lead Wires			Application
			Standard Length	Construction	Colour	
Instantaneous (Aluminium and Copper tube)	-	-	1.8m	Tinned steel PVC coated	Yellow for copper tube and blue for aluminium tube	Copper tube-underground coal mines. Aluminium tube for other electric blasting operations.
Long Delay (Aluminium and copper tube)	0-10	300ms	3m	Tinned steel PVC coated	One wire yellow & the other green	Tunnelling, shaft sinking, drifting etc.
Short Delay (aluminium and copper tube)	0-10	25ms	3m	Tinned copper/steel PVC coated	Both wires green	Quarrying/open cut excavations, stoping, tunnelling, shaft sinking, drifting etc.
Carrick (non-incendive) Short Dealy (copper tube)	0-6	25ms	2.5ms	Tinned copper/ tinned iron PVC coated	Both wires orange	Delay blasting of coal off the solid.

Table 17 (continued)

Delay detonator Relays	-	17ms	-	-	-	Quarrying/open cut excavations.

SPECIFICATIONS AND FIRING CHARACTERISTICS

Electrical Resistance			**Firing Characteristics**		
Length of Lead Wires (m)	Tinned Steel (24 SWG)	Tinned Copper (23 SWG)	i.	Firing impulse :	4-6 milli-watt seconds/ohm
1.8	2.5-4.1 ohms	1.9 ohms (maximum)	ii.	Minimum single firing current :	0.50 amps. (Current application time 50 milliseconds)
2.5	3.3-5.0 ohms	2.1 ohms (maximum)	iii.	Series firing current :	1.25 amps. (Current application time 4 milliseconds)
3.0	3.6-5.7 ohms	2.2 ohms (maximum)	iv.	'No-Fire' current :	0.30 amps. (Current application time 50 milliseconds)

ANFO, LOX, Slurry Explosives. Emulsion Explosives, Heavy ANFO, etc.

Question 32 : Discuss briefly about the plain detonators and the electric detonators.

Answer : Plain Detonators (Non-Electric type):

The plain detonator contains a base charge of pentaerythritol tetra nitrate (P.E.T.N.) with a priming charge of ASA composition (lead azide, lead styphnate and aluminium powder) in an aluminium tube. The ASA initiates the base charge of P.E.T.N. It is insensitive to shock, friction and lightning.

Electric Detonators

Electric detonators are made with either aluminium or copper lubes and are fired by passing electric current through a fusehead. A current of 0.5 amp at a minimum 3.5 volts can ignite the fusehead composition which in turn initiates the priming charge. These are essentially of two types:

- Instantaneous detonators (Fig. 7)
- Delay detonators (Fig. 8)

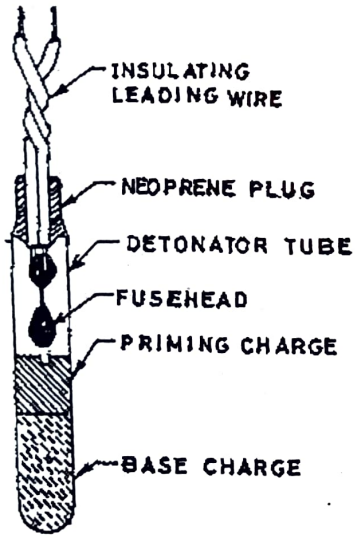


Fig. 7 Sectioned Electric Detonator (Not to scale)

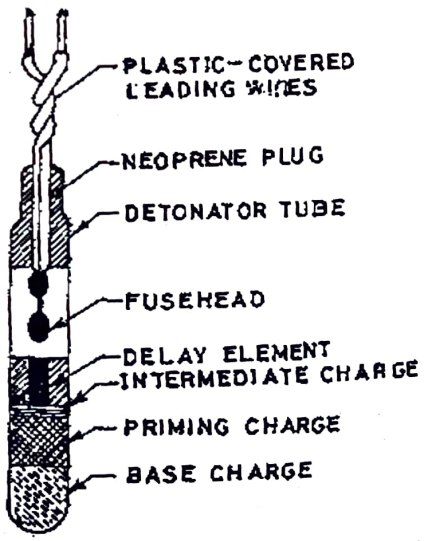


Fig. 8 Sectioned Delay Detonator (Not to scale)

The base charge and the priming charge contained in the detonator tube are the same as in a plain detonator. The fusehead assembly for electric ignition of the charge consists of an LMNR/Chlorate fusehead fitted with PVC covered tinned iron or copper leading wires and a

neoprene sealing plug. The fusehead assembly is directly crimped onto the detonator tube in the case of instantaneous detonators but in the case of delay detonators, a suitable delay element consisting either copper or brass sleeve filled with a special composition is inserted between the fusehead and the priming charge. The delay is resulted by varying the length of the sleeve containing the special composition which burns at a specified rate.

Delay detonators are manufactured in two different series:

- Short Delay (Milli-second delay detonator)
- Long Delay (Half-second delay detonator)

The delay detonators are the low tension electric detonators.

In the short delay series, a delay interval of 25 ms has been provided between the subsequent delay nos. whereas the interval in the case of long delay series is 300 ms. In underground coal mines only copper detonators should be used. Aluminium detonators are not permitted.

In the surface mining blasting operation the legwires of the detonator must be long enough, one end of it must reach upto the primer into the blasthole. The blastholes are fired by the delay detonators. Upto 25ms comes under the short delay range and 50ms to 100ms comes under longer delay range.

Question 33 : What merits are obtained by using the delay detonators?

Answer : The chief merits of using delay detonators are as following :

- Due to blasting with delay detonators more efficient blasting is possible because of the availability of free face and hence consumption of explosive will be less. For example, when number 2 detonator is blasted it will create a free face so that the tendency of blasting effect by the no.3 detonators will be towards that created free face.
- The short delay detonator (millisecond delay detonator) produces less ground vibration,
- The whole round of shots can be fired in a fraction of second which saves considerable time (i.e., time for repeated inspection, charging and connections, clearance of fumes and gases, etc. are reduced).
- Due to the collision of blasted rock resulted from the

successive shots the amount of fragmentation of rock will be increased.

Question 34 : Which detonators are much safer to use? What problems are associated with the firing of electric detonators?

Answer : Starting with safety fuse and ordinary detonators, it had been considered that electric detonators were much safer and more modern. However, electrical firing involves many problems of detonators and extraneous electricity hazards (such as stray current, static electricity, atmospheric electricity, radio frequency, high voltage power lines, etc). In addition there are severe demands on the exploders. With electric delay detonator firing time can be controlled and delay period will be accurate.

Question 35 : Discuss briefly about various types of detonators.

Answer : The various types of detonators which are manufactured by the IDL Chemicals Ltd. is discussed briefly in Table 18.

Question 36 : Write short notes on the following :

(a) Insensitive detonators (b) Electronic detonators (c) Magnadet detonators.

Answer :

a) Insensitive detonators

Here insensitive characteristics is given by designing a matchhead which requires a large electrical impulse to initiate it. So these types of detonators are very safe to use where high electrostatic fields are likely to be present, which may cause initiation of any other type of detonators and result in accident.

b) Electronic detonators

Electronic delay detonators are based on micro-chip technology. The electronic detonator is basically a device which stores electrical energy for a certain time and then delivers that energy as a sharp pulse at a precise time to a conventional blasting cap or electric blasting circuit. With these types of delay detonators possibility of overlap is completely eliminated. Static electricity, radio frequency or stray currents have no influence on it.

c) Magnadet electric detonators

This type of detonators are initiated electrically and use the principle of transformer coupling to provide electric insulation. These

Table 18

Type		Characteristics	Application	Specification
1.	Special ordinary detonator (Fig.-9)	Primary charge consists of ASA & PETN as primary and secondary charge. The spit of fire conveyed by the safety fuse is picked up by the primary charge, which detonates the secondary charge for initiating high explosive charge.	It's high strength ensures complete initiation of cap sensitive slurry explosives. It can be used in work sites having potential static hazard, near high power transmission lines, etc., for shallow hole blasting in both opencast and underground mines, project construction sites, etc.	Shell—Aluminium tube Strength—No. 8 Shell dimensions—Length—37mm Inner dia—6.4 mm Outer dia—7mm
2.	Instantaneous Electric Detonator (Fig.-10)	(a) The shells manufactured in Aluminium filled with ASA and PETN as primary and secondary charge. Electrical fuse heads are fitted to the filled shells. PVC extruded legs wires are sleeved through a PVC plug and crimped to the detonator to ensure protection against moisture & water. The led wires are water proof rugged by abrasion resistant, easy to strip and handle.	They can be used in quarrying, opencast and underground mines and on construction sites but they can not be used in underground gassy coal mines.	Shell—Aluminium tube Strength—No. 8 Leg wire colour—white Shell dimensions—Length—37mm, dia—7mm Leg wire diameter—24 SWG iron wire Leg wire resistance—0.55± 0.15 ohms/m Fusehead resistance—1.6 to 2.0 and 2.0 to 2.4 ohms Firing impulse—3.2 mw seconds per ohm Minimum firing current per series—1.2 amperes D.C. 'No-Fire' current—0.18 amperes applied for 300 seconds.

Table 18 (continued)

	Type	Characteristics	Application	Specification
		(Instantaneous electric detonators are manufactured in copper coated steel tubes. Other details are as same as mentioned in 2(a). Fig.-7	They can be used in underground coal mines.	Shell—Copper coated steel tube. Strength—No. 6 Leg wire colour—white Shell dimensions—Length—37mm Leg wire diameter—24 SWG iron wire. Leg wire resistance—0.55 ± 0.15 ohms/m Fusehead resistance—1.6 to 2.0 and 2.0 to 2.4 ohms Firing impulse—3.2 mw seconds/ohm. Minimum firing current per series—1.2 amperes D.C. 'No-Fire' current—0.18 amperes applied for 300 seconds.
3.	Long Delay Electirc Detonator Fig.-8	It consists of an slumimium tube filled with ASA and PETN as primary and secondary charge. These detonators have delay number ranging from zero to ten with a nominal time interval of half second between each successive delay detonator.	Judiciously usage of these detonators achieves good fragmentation with a controlled throw thereby enabling easy mechanical loading. They also provide greater control of vibration, noise and fly rock. They can be used for driving drifts and tunnels, blasting raises, sinking shafts, stoping in metal mine, trenching and other tight blasting cinditions. They can not be used where methane or coal dust occur.	Shell—Aluminium tube Strength—No. 6 Leg wire colour—Red/red Shell dimensions—varies for different delay nos. Leg wire diameter—24 SWG iron wire. Leg wire resistance—0.55 ± 0.15 ohms/m Fusehead resistance—1.6 to 2.0 and 2.0 to 2.4 ohms Firing impulse—3.2 mw seconds per ohm Minimum firing current per series—1.2 amperes D.C. 'No-Fire' current—0.18 amperes applied for 300 seconds.

Table 18 (continued)

	Type	Characteristics	Application	Specification
4.	Short Delay Electirc Detonator Fig.-9	It consists of an slumimium shell filled with ASA and PETN as primary and secondary charge. These detonators have delay number ranging from zero to ten with a nominal time intercal of 25 milliseconds between successive delay number from 1 to 6, 50 milliseconds for 7 and 8 and 75 milliseconds for 9 and 10.	Judicious usage of these detonators achives good fragmentation with a controlled throw, enabling easy mechanical loading. They also provide greater control of vibration noise and fly rock. They can be used in opencast and underground metal mines, for driving tunnel, drifts and shaft sinking and in quaries where multi shot rounds are required. They can be used in both dry and wet conditions but not where methane or coal dust occur.	Shell—Aluminium tube. Strength—No. 6 Leg wire colour—Blue/red Shell dimensions—varies for different delay nos. Leg wire diameter—24 SWG iron wire. Leg wire resistance—0.55 ± 0.15 ohms/m Fusehead resistance—1.6 to 2.0 and 2.0 to 2.4 ohms Firing impulse—3.2 mw seconds/ohm. Minimum firing current per series—1.2 amperes D.C. 'No-Fire' current—0.18 amperes applied for 300 seconds.
5.	Coal Delay Detonator Fig.-10	It consists of a copper coated steel shell with ASA and PETN as primary and secondary charge. Coal delay detonatiors are nonincendive in nature and have delay number from 0 to 6 with a nominal time interval of 25 milliseconds between each successive delay number.	When used properly coal delay detonators give desired advance with good fragmentation controlled throw and reduced vibrations. They can be used for all types of blasting in underground coal mines. They can be used in dry, damp or wet conditions and also where methane or coal dust occur.	Shell—Copper coated steel tube. Strength—No. 6 Leg wire colour—Red/white Shell dimensions—varies for different delay nos. Leg wire diameter—24 SWG iron wire. Leg wire resistance—0.55 ± 0.15 ohms/m Fusehead resistance—1.6 to 2.0 and 2.0 to 2.4 ohms Firing impulse—3.2 mw seconds per ohm Minimum firing current per series—1.2 amperes D.C. 'No-Fire' current—0.18 amperes applied for 300 seconds.

types of detonators are protected from all sources of extraneous electricity for initiation and need at least 25 kHz frequency current. These types of detonators are very simple, reliable and available in both short delay and half-second delay range. Special exploders operated by battery or operated by mains and also multichannel machines are available to initiate these Magnadet type electric detonators. The system provides adequate safety from radio frequency radiation and also from static hazards when the holes are pneumatically loaded with ANFO.

Question 37 : Discuss firing system with the Electronic Delay Detonator.

Answer :

Electronic Delay Detonator Firing System

This system comprises the following :

(1) upto 200 integrated electronic detonator firing caps,

(2) a console for the testing and programming of each integrated electrinic detonator (IED),

(3) a bifilar line which links the firing console to the firing stand,

(4) a firing console for operational check and the management of the firing itself.

An electronic circuit which is programmable from 1-6000 ms in 1 ms increment generates the delay timings.

Except incorporating a slightly lengthened version of the current PVC plug the detonator looks identical to an instantaneous detonator.

Firing of these type electronic detonators are done by manually and computerized method.

In manual firing system each IED firing cap is programmed individually using the programming console which records their order number and corresponding delay. The detonators are then linked in parallel to the firing console which is connected to the programming console. The firing console then automatically checks the integrity of all the electrical connections, the efficacy of each IED firing cap and the proframmed delay timings. The operator then with the help of firing fire the electronic delay detonators.

In case of computerised firing system each electronic detonator receives an order number using the programme console before detonators are connected in parallel to the firing console. The pre-determined firing

pattern is then transferred from a personal/lap-top computer to the firing console after assigning automatically the pre-set delay timings to the respective IED. The firing console then automatically carries out the integrity check and thereafter fire the circuit when the blaster wanted to do so.

The Electronic Delay Detonators Offer the following Advantages :

- offer flexibility and reliable sequencing,
- provide accurate timing, reduce ground vibration, improve fragmentation, enhance selectivity, reduce blast orineted damage,
- reduce operating cost,
- improve delay precision,
- offer re-programming of delays of detonators at any time till the moment before firing,
- offer automatic checking facilities,
- offer facilities of easy, quick and most accurate way of preparing a blast for firing a complex circuit,
- reduce over-braking problem,
- offer much safety in initaition process.

Question 38 : Discuss briefly about the various causes of hazards and control measures in firing by an electric detonator.

Answer : The various causes of hazards associated in firing by electric detonator and their controlling measures are as follows :

Static electricity : The static electricity is generated during loading slurry explosives, ammonium nitrate, etc. by the tubes constructed by the non-antistatic material. Even storm can generate static electricity due to friction among the sand particles. This static electricity when reaches more than the ignition impulse of the detonator, initiates the detonator and explodes the charge around it. So with the use of tubes for loading the charges or stemming material constructed with the antistatic material, stopping of charging operation during stormy weather and the use of modern detonators immune to static ignition will be the remedial measures against it.

Lightning : Lightning in a cloudy weather or in a rainy day may

cause premature initiation of the detonators by the strong electric field. During lightning a very high voltage will be generated inside the rock and also a very high voltage will be developed in between the detonator and its lead wires if the latter touches with any conductive material like pipe lines, rails, etc. and may cause initiation of the detonator. As a safeguard against the lightning surge the wires should be rolled and placed in the hole and evacuation of the blasting site whenever a thunder is heard from a great distance, etc. will solve the problem.

Current leakage due to insulation fault : The leakage of current is possible due to the insulation faults of the electrical equipments and rails, pipe lines and other conducting materials may acquire electric tension. If the wire leads of the electric detonator at any time came into contact with them, the detonator may be initiated. So during drilling or charging operation no detonator wire leads should come into contact with any metallic conductor like rails, pipes, etc.

Radio frequencies : Although in normal situation the radio frequencies do not have any significant effect on blasting but in abnormal condition radio waves may cause initiation of the detonators provided that the distance between the wires of the blasting circuit and the antenna is very close. So as a safeguard the distance between the wires of the blasting circuit and the antenna should be kept beyond the safe distance limit which depends upon the power of the transmission station, wave length, etc. and may vary from a few hundred metres to a few kilometres.

Power lines : The power lines also generate ground potential which if reaches sufficient values may ignite detonator. So wires or joints of the blasting circuits should not be within a distance of 50 to 60 metres from the power lines depending upon the voltages of power lines and the types of the detonators used.

Question 39 : What are the main reason of current leakage in the blasting circuit? What precautionary measures are to be taken against it?

Answer :

Current leakage in the blasting circuit

The leakage of current in the blasting circuit is owing to the following reasons :

- damp wet ground,
- insulation damage in the leg wires,
- high conductivity of the mineral/rock body,
- presence of water in the blastholes,
- poor insulation between bare wires at splices and ground.

Precautionary measures

The following precautionary measures may be taken against the leakage of current :

- avoid splicing of any blasting cord/wires in the blastholes,
- avoid any connection touching the ground,
- fire optimum number of shots per round depending upon the design of the exploder/blasting machine and also use the appropriate type of blasting machine,
- avoid using of lighter gauge lead-line since it will supply less current in the blasting circuit,
- avoid damaging of any leg wires during loading of charge into the blastholes.

Question 40 : Briefly write about the following :

a) Detonating fuse. With the help of neat sketches show the various joints in the detonating fuse.

b) Primex boosters.

c) Detonator relays. Use of delay detonator relays.

Answer :

a) Detonating fuse

Like detonators it also initiates high explosives, but for initiating a detonating fuse a detonator is needed. The detonating fuse resembles like a flexible tube, the centre of which conists a cord of high explosive like PETN to detonate the explosives around it. In long hole blasting in which case the whole length of the column charge comes into the contact of the detonating fuse, the blasting (either single blasthole or multiple shot with sequence blasting with the help of detonator relays) with the help of detonating fuse is very much useful.

Example : Cordtex Detonating fuse

It is used for initiation of cap sensitive explosives. It is initiated by a No.6 detonator and detonates with a velocity of approximately 6500-7000 m/sec. It consists of a PETN core enclosed in a tape which is wrapped with cloth. The fuse is then totally enclosed in a tubular plastic white material cover; whose external diameter is around 5mm with an average weight of about 16-20 gm per metre length. It can be used under watery conditions since it is wrapped with the plastic cover but to use in a watery mine the end of the fuse should be sealed with tape or water proofing material. Detonating fuse connected to a large number of shots can be blasted by a single detonator. Non incendive detonating fuse like 'RINGCORD' is used for blasting in the ring holes in the blasting gallery method of working.

Fig. 15 shows the various joints used with the detonating fuse.

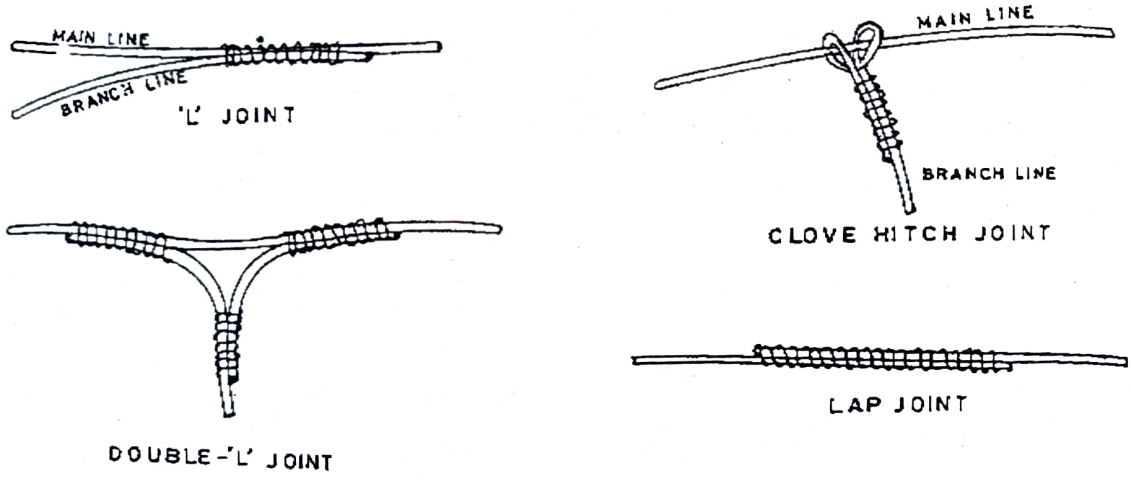


Fig. 15 Various joints in detonating fuse

b) Primex Boosters

It is a cast booster charge for non-cap sensitive explosives. Initiated by Cordtex detonating fuse it detonates at about 7000 m/sec. They are available in the sizes of 23, 37 and 50mm dia with weights of 20, 100 and 250g respectively.

c) Delay Detonator Relays

In non-electric delay firing system for initiating explosives, detonating relays are used in the detonating fuse. It is basically two open ended delay detonators having neoprene sleeve in the middle and enclosed by aluminium sleeve tube which can be readily crimped into a 'cordtex' detonating fuse line. The delay interval for each detonating

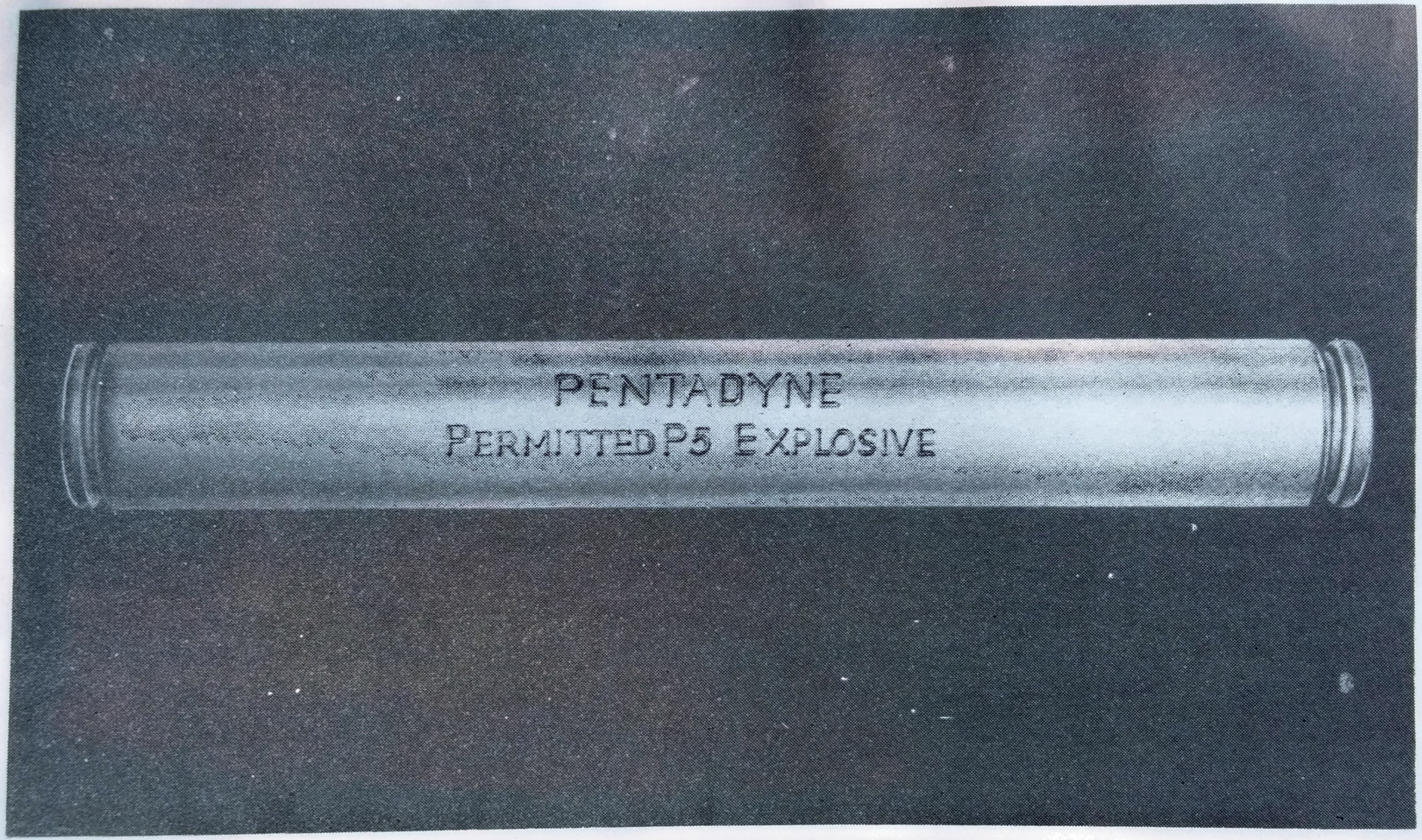


Fig. 6 Pentadyne Permitted P_5 Explosive, *by Courtesy* : IDL-Chemicals Ltd.

Fig. 9 Special Ordinary Detonator

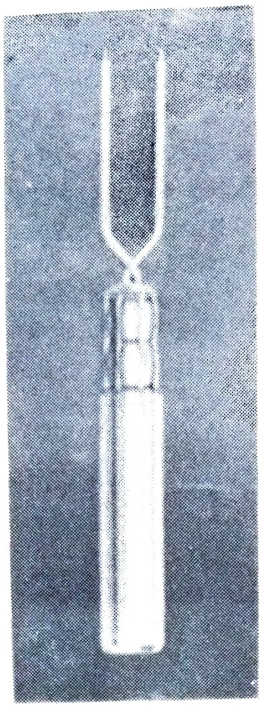


Fig. 10 Instantaneous Electric Detonator.

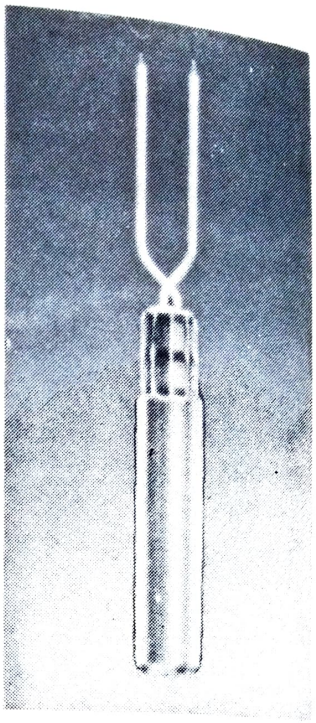


Fig. 11 Instantaneous Electric Detonator for use in Underground Coal Mines

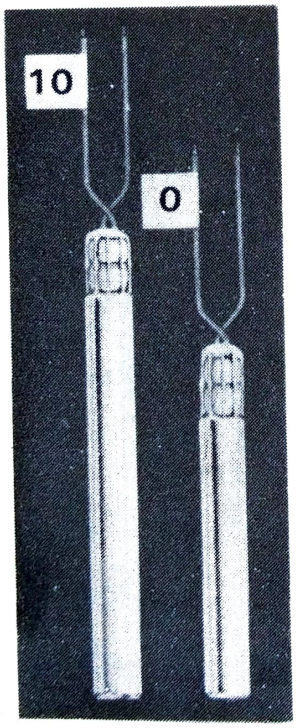


Fig. 12 Long Delay Electric Detonator

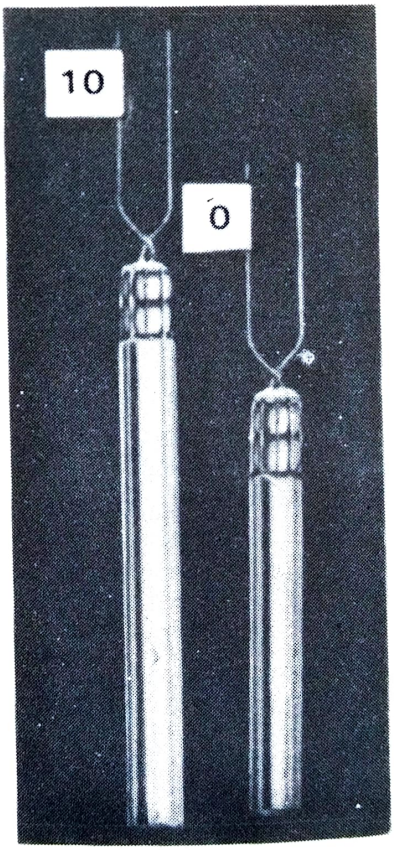


Fig. 13 Short Delay Electric Detonator

All the illustrations of Detonators and Booster
by Courtesy : IDL-Chemicals Ltd.

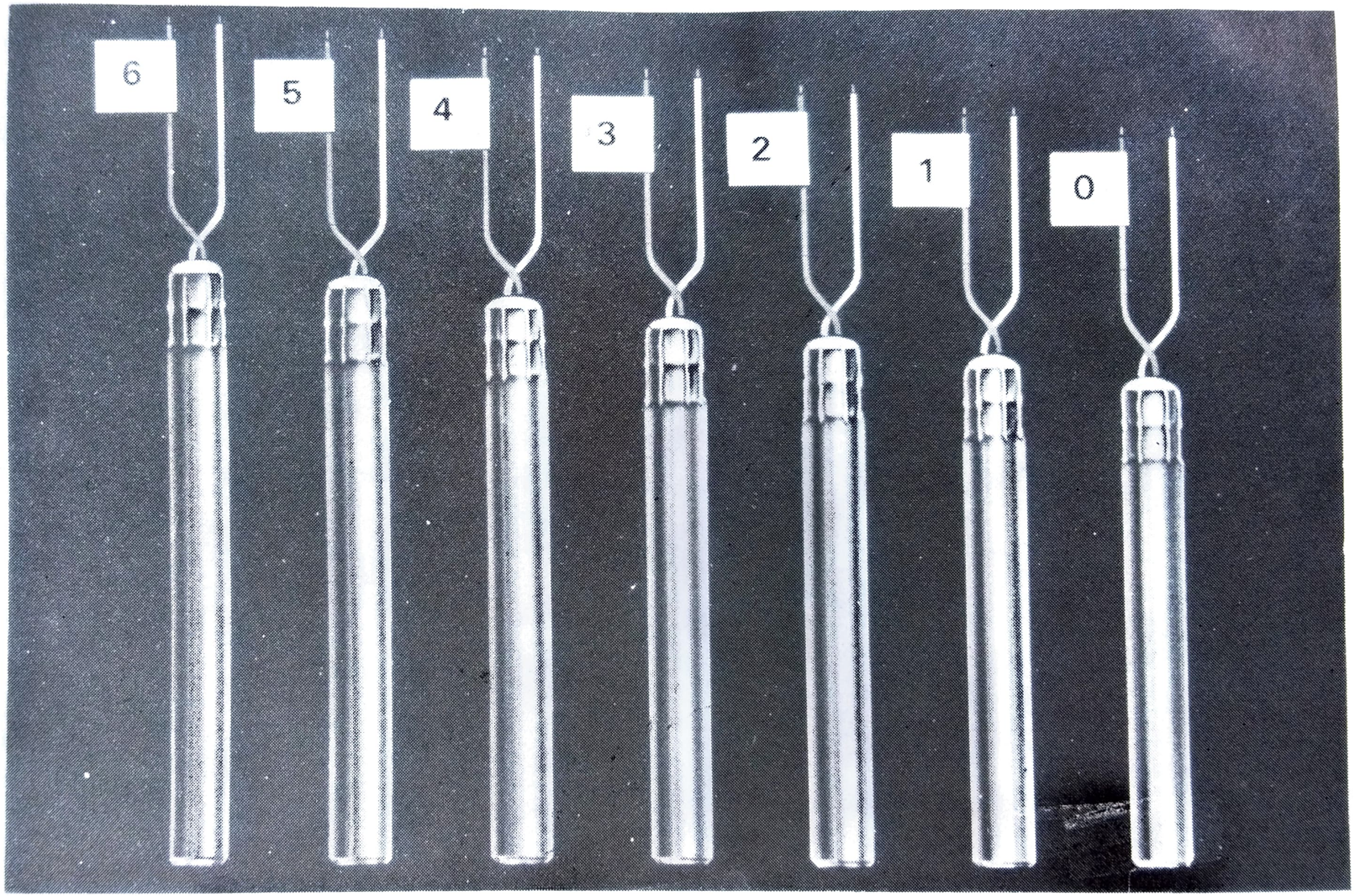


Fig. 14 Coal Delay Detonator, *by Courtesy* : IDL-Chemicals Ltd.

Fig. 17 Pentolite Booster
by Courtesy : IDL-Chemicals Ltd.

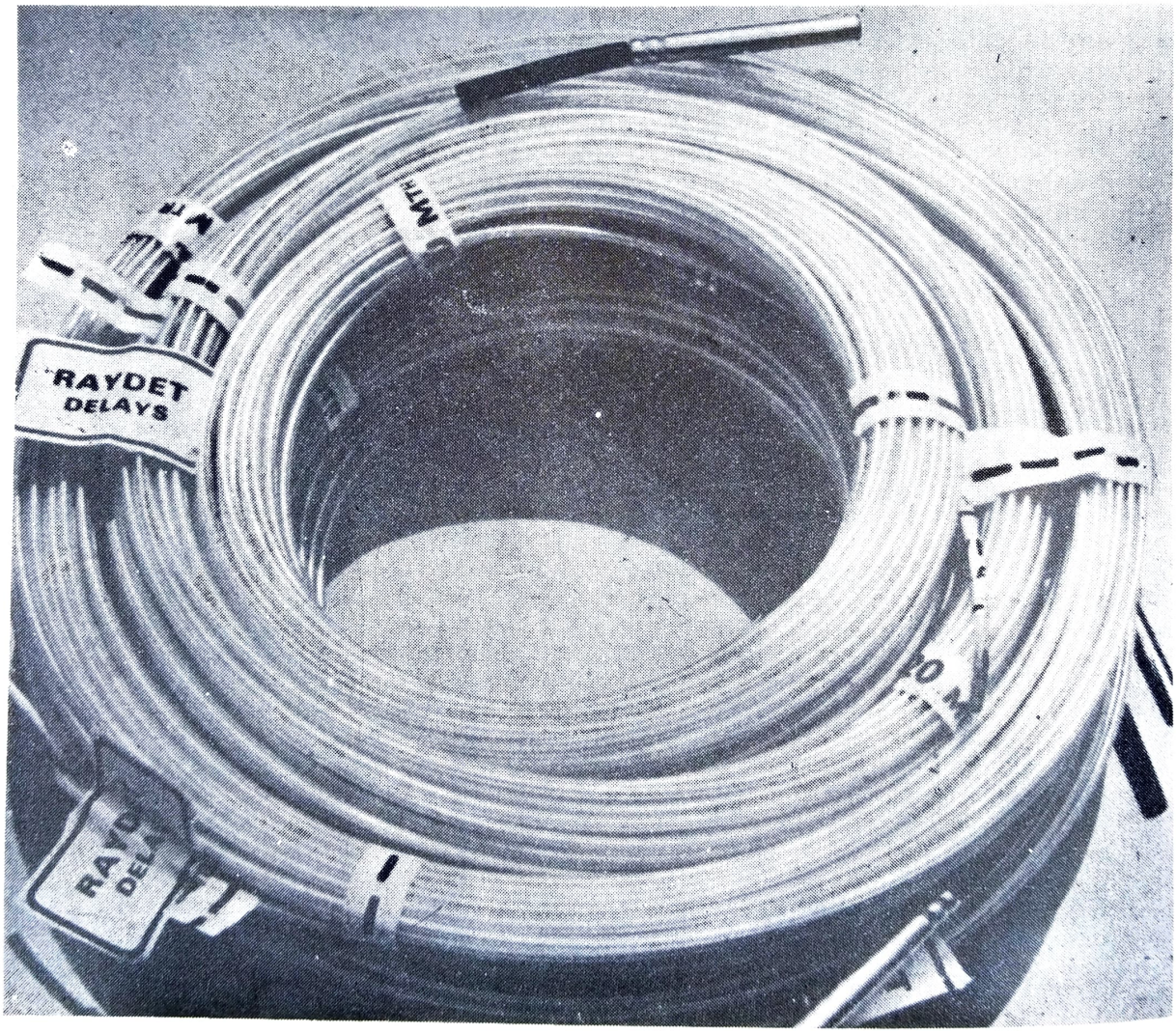


Fig. 18 Raydet, *by Courtesy* : IDL-Chemicals Ltd.

relay may vary from 15 to more than 100 milliseconds. Since non-electric in nature, they are insensitive to stray currents and static electricity and hence they can be effectively used during heavy storm, cloudy weather or on a rainy day when problem of stray currents and electrical storm may arise. It also gives better fragmentation and powder factor and eliminates the use of heavy amount of electrical detonators in the circuit (only one detonator is required to fire a complete circuit). The lower delay interval detonator relays are used for smaller diameter holes whereas higher delay interval detonator relays are used for larger diameter holes. The detonator relays should never be used inside the blast hole.

For use with cordtex detonating fuse a nominal firing interval of 17 ms delay detonator relays (Fig. 16) are generally used.

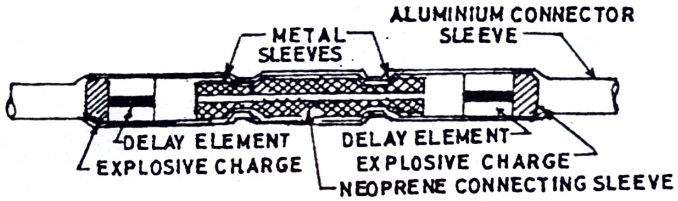


Fig. 16 Diagram of detonating relay

Question 41 : Discuss briefly about the pentolite booster.

Answer :

Characteristics

Pentolite boosters make efficient primers and boosters that detonate at a very high velocity and temperature. These qualities, together with high density produce great detonation pressure. Pentolite boosters can be stored for long periods of time, as they contain no liquid ingredients and can be operated easily without use of prickers. They are equipped with two holes for easy initiation with detonating cord or detonators. They do not contain nitro-glycerine or other headache causing ingredients (Fig. 17).

Applications

Pentolite boosters can be used for effectively initiating cap-insensitive charges in a borehole, at any predetermined point in a column of explosive charge.

Specifications :

Density	1.55 to 1.61 gm/cc
Velocity	7800 metres per second

Shock energy	237 calories/gram
Bubble energy	470 calories/gram
Detonation pressure	240 kilobars

Question 42 : Discuss about the detonating cord after giving some examples.

Answer : A 'Detonating cord' is a flexible cord containing a uniform core-load of PETN that ensures positive initiation of cap-sensitive explosives, along the entire length of fuse. The high explosive core is enclosed in several textile coverings and plastic to protect the PETN core from mishandling, exposure to extreme temperatures and humidity as well as adverse mining conditions. It is insensitive to detonation by friction or shock as met with under normal conditions of usage and requires a proper initiation for detonation.

Effects of Detonating cord over the Booster Sensitive Explosive on initiation are as follows :

- the shock energy phase of the explosives degrades at a very faster rate compare to the heave energy of the explosive charge.
- effects adversely the explosive energy characteristics.
- desensitise a little the booster sensitive explosive. The loss of energy depends upon the factors tike diameter of the blasthole, core load of the detonating cord, type of explosive, etc.

Therefore except in some specialised applications, point initiation system of the explosive charge is the best proposition.

Applications and specifications

(A) *Manufacturer :* Maharashtra Explosives Ltd.

Type	MEXCORD-II
Nominal Diameter	4.8 mm
Nominal core-load	10 gms/m
Average velocity of detonation	6500 m/sec
Initiation	By No.6 detonator
Colour	Pink
Application	Can be used under all conditions in Mining, Quarrying and Construction Works.

(B) *Manufacturer :* IDL Chemicals Ltd.

Table 19 (Explosive Manufacturer IDL Chemicals Ltd.)

	D-CORD I	**D-CORD II**	**D-CORD III**	**L-CORD II**
Nominal Diameter	3.9 mm	4.65 mm	6.0 mm	3.0 mm
Approximate weight per metre	17 gms	21.5 gms	35 gms	
Weight of PETN per metre	6 gms	10 gms	20 gms	3.5 gm
Average velocity of detonation	6800 mtrs/sec	6800 mtrs/sec	7000 mtrs/sec	6500 mtrs/sec.
Initiation	No. 6 detonator	No. 6 detonator	No. 6 detonator	No. 6 detonator
Colour	Yellow	Orange	Red	Blue
Applications	Surface trunklines, secondary blasting & downlines in small dia holes	Excellent general purpose cord for reliable blast, initiation under virtually all conditions in mining, construction & quarrying	Special applications such as siesmic prospecting, pre-splitting, axial priming, etc.	Downline/ Trunkline

(C) *Manufacturer :* Indo Gulf Explosives Ltd.

AVICORD II

AVICORD-II is a flexible detonating cord manufactured by IGEL and contains PETN in its core and is used to transmit a detonation wave from one system to another. It is insensitive to detonation by friction or shock and requires a proper initiation.

Applications : AVICORD-II can be used under all conditions of mining, Quarrying and Construction.

Specifications :

Overall Nominal Diameter 5 mm ± 0.2 mm

Weight of PETN per metre	10 gms ± 0.2 gms
Average velocity of Detonation	6500 m/scc ± 500 m/sec
Water Resistance	Excellent
Initiation	No.6 Detonator
Flexibility	Flexible upto a minimum temperature of -20° C and maximum of 60° C
Shelf Life	One year
Packaging	Packed in wooden reels each of 375 metres. Four reels are packed in a case to give 1500 metres of AVICORD-II detonating fuse per case.

Detonating Cord Developed by IES

Types of the detonting cord	CORDTEX	CORDFLEX
Nominal diameter, mm	5.0	4.5
VOD, m/s	6500	6500
Nominal weight of PETN, g/m	10	7
Initiation	By No.6 detonator	By No.6 detonator
Colour	White	Blue, Light green
Water resistance	Excellent	Excellent
Nominal tensile strength, kg	70	60
Category	Class-VI Divn.-II	Class-VI, Divn.-II
Application	In the quarries, opencast mines, underground metalliferous mines, under-water blasting etc.	In the quarries, opencast mines, underground metalliferous mines, surface excavation etc.

Presently **lower energy detonating fuses** are developed. The amount of explosives in it varies from 1.6 gm/m to 8 gm/m length. The lower energy detonating fuses which are presently being used in the opencast mines, contains explosives at a amount of 5 gm/m length.

Lower energy detonating fuses improve fragmentation, reduces explosive quantity, noise and vibrations.

Question 43 : Discuss briefly about the 'CORD RELAY'.

Answer : Cord Relay is a non-electric surface delay initiator used in conjunction with detonating cord trunklincs. Cord Relays are bidirectional in usage but uni-directional in application. The plastic housing used for placing the delay units is rugged, streamlined and simple in design for easy hook-up during usage. Holes are provided on either side of plastic housing for hook-up of detonating cord. Cord Relays are accurate and available in delay timings of 17, 25, 50 and 100 millisecond intervals.

Cord Relays provide an accurate delay between holes in the same row or between rows of holes as desired. Infinite number of rows can be blasted by placing Cord Relays between rows in sequence, thus helping the blasting engineer to carry out non-electric sequential blasting. A delay interval commensurate with burden can be provided by connecting Cord Relays of different delay intervals in series in the hook-up. This flexibility is not possible with electric delay detonators except perhaps to some extent with Sequential Blasting Machine (SBM). Use of millisecond delay detonators with conventional exploders greatly restricts the blast size since only 10 delay periods are available and in order to provide adequate delay interval between rows, it would be necessary to skip a few delay numbers in between numbers 0, 2, 4, 6 or 0, 3, 6, 8 etc., thus restricting the maximum number of rows that can be blasted with delay detonators. With Cord Relays, any delay interval can be provided and this can be progressively increased for the back rows for better relief. For large burdens such as those used in Coal overburden and Iron ore mines (8-10m bench height) 100-125 millisecond delay interval can be provided without fear of cut off and has been successfully used in blasts.

Judicious placement of Cord Relays also helps in controlling fragmentation and direction of throw. Initiation patterns can be so designed that ore/waste, high grade/low blocks can be blasted to form separate heaps. Where required, blasts can be designed in such a way that only one drill hole blasts at a time thus controlling ground vibrations. In Cord Relays, the blasting engineer has a very inexpensive but powerful tool. Fig. 18 (a) shows the security of detonating cord in the cord relay by the double loop system.

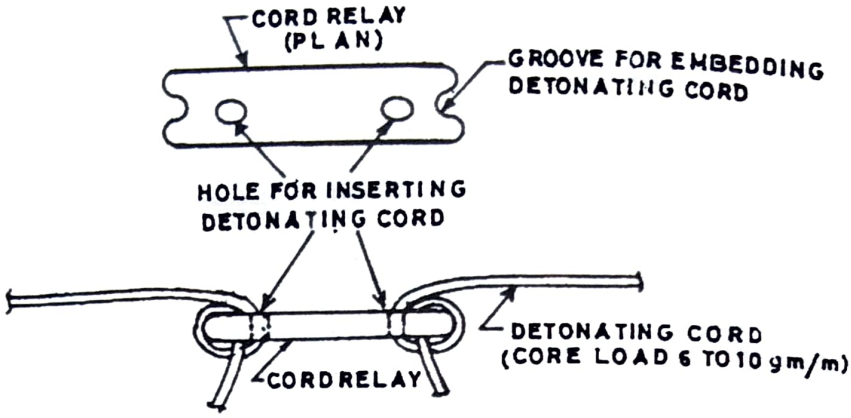


Fig. 18 (a) Securing of detonating cord in the cord- relay by the double loop system

Advantages

Safe	Cord Relays are non-electric and are thus immune to extraneous electricity sources such as stray currents, static electricity and radio frequency (RF) energies.
Misfire free	Use of tie-lines in the surface hook-up provides two or three detonation paths which minimises risk of misfire.
Simple	Cord Relays are bi-directional in usage and very simple to use
Accurate	Cord Relays are accurate. They get initiated sequentially and thus eliminate problems of scattering prevalent in delay detonator.
Unlimited	Firing of infinite number of holes are possible. They provide an unlimited number of delays allowing large blasts to be conducted.
Less vibrations	Use of combination of Cord Relays of different timings allow large blasts to be carried out with minimum ground vibrations.

Firing of one hole at a time is possible.

Instructions for usage

1. Lay the detonating cord trunkline with minimum slack between holes or row of holes.
2. Cut the trunkline at the point where Cord Relay is to be hooked-up.
3. Pass one cut end of detonating cord through one hole provided in the relay. Loop the cord and tie a knot or fasten with tape.
4. Repeat the procedure at the other end.

Question 44 : What is the degree of coupling, prills, sequential firing, igniter cord, misfire?

Answer : Degree of coupling: It is the measure of extent of direct contacts among the explosive charge and the wall of the blast hole. With good coupling chances of loss of energy through the space between the explosive charge and the wall of the blast hole will be very less.

Decoupling : It is achieved when the diameter of the explosive is less than the diameter of the blasthole. Air-decking on the other hand involves the use of fully coupled explosives for only a fraction of length of the blasthole with a column of air or other inter material between the explosive charge and the stemming column. The extent of reduction of peak blasthole pressure depends on the degree of charge decoupling which ultimately reduces the peak blasthole pressure to a levels equal to the tensile strength of rock. The peak blasthole pressure ^{P}bhole for a decoupled charge can be determined from a knowledge of coupling ratio f_0 (defined as the ratio of charge volume to borehole volume), the explosive density ($^{\rho}e \times p$) and the VOD of the explosive, the peak borehole pressure can be expressed as

$$^{P}\text{bhole} = 0.25 \int_{f_0}^{1.2} \rho\text{exp.VOD}^2$$

Prills

When concentrated ammonium nitrate solution is sprayed against a stream of air, a cellular sub-globular particles of ammonium nitrate is formed which is termed as prills.

Prilled ammonium nitrate is used for -

(a) higher thermal stability and coating : for free flowing during transportation, storage and handling

(b) low density : for optimum volumn build-up/charge distribution in the boreholes for uniform fragmentation and higher yield factor

(c) porosity : for optimum oil absorption and maximum energy output

(d) optimum crushing strength and wear resistance that prevent prill breakdown during transportation/storage/handling and machine fines generation.

Sequential firing

It is a system of firing in which the charged holes are initiated progressively in the scale from the lower resistance to higher resistance holes to reduce the burden on each subsequent holes in turn. This will result in creating of free faces for the each subsequent hole and will give better yield. It is done by firing with safety fuses or initiating with electric delay detonator or detonating cord.

Igniter Cord

It is a cord having some ignition compound used for lighting any number of safety fuses in any desired sequence. When it is ignited, it produces intense flame and progress along its length at a uniform rates. When it reaches each connector (metal igniter cord connectors are usually pre-crimped to the free ends of the safety fuses) it automatically lights each safety fuse subsequently in turn. However, length of the safety fuses should not be less than 1.2 m. Faster type igniter cord can burn at the rate of 5 m/min (approximately).

Misfire

If a charge or a part of the charge in a blast hole due to any reason does not explode, it is termed as misfire. Any misfire, if occured, is to be dealt with very carefully by a competent person following the instructions stipulated for the purpose. Misfires happen due to many reasons like defective explosives, detonators, discontinuities in the electrical circuit, defective exploder, high resistance of detonator leads, insufficient current in the circuit, cut offs, damage of any of the blasting components while charging the blast holes, etc.

Question 45 : What is a safety fuse?

Answer : It looks like a cord and it contains a core of finely grained gunpowder wrapped with layers of tape or textile yarn (to provide abrasion resistance to the fuse) and externally it is coated with plastic sheathed material or waterproof coatings to protect it from moisture and water. If one end of the fuse is ignited, it carries a flame with a burning speed at the rate of 100 to 120 sec/metre and either ignite gunpowder or detonate an ordinary detonator inserted in the cartridge itself which subseqently detonates a high explosive. When igniter cord is used to ignite the safety fuse the metal igniter cord connector should be crimped to the free end of the safety fuse.

Question 46 : What is the purpose of a circuit tester and a crimper?

Answer : To check the soundness of the circuit for ensuring that there is no open or short circuit connected by the leads of the electric detonators approved type circuit testers are used. The current passed through the circuit tester should be limited to minimum to avoid accidental explosion and also the test is carried out from a safe distance and place from the site of blasting. Beside above, circuit testers are used to check loose connection, missed connection, leakage to ground by a bare wire from broken insulation, current leakage from extraneous sources (stray electricity), etc. A blasting galvanometer having a silver chloride cell with high resistance coils attached in series incapable to initiate any detonators when in operation, is used to measure many of the above parameters. For measuring stray current, voltage, resistance, current in milliamperes, etc. blasting voltohmmeter may be used. As a safeguard all the testing instruments mentioned should be checked regularly by a competent person.

A crimper presses the ends of a plain detonator tube on a safety fuse inserted into it to control the fuse end not to come out from the detonator.

Quation 47 : How are resistance of circuits chacked with the Blasting Ohm metre?

Answer :

Checking resistance of circuit with the Blasting Ohm meter

- First of all short the blasting ohmmeter by placing piece of wire or screwdriver across the terminals. If necessary adjust the adjusting screw in such a manner that the needle of the Ohmmeter shows the zero reading.
- Unshunt the electric detonator's legwires and touch each leg wire's end to a terminal of the Ohmmeter. Find out the resistance of the detonator.
- Separate the bared tips of the leadline at both ends. Touch the tips at one end to the Ohm meter's terminals. The needle must be on the zero position otherwise there is a short circuit in the leadline. (Fig. 19)
- Next shunt the bared wires at one end of the leadline and the tips of the other end shall be touched with the Ohm meter. The

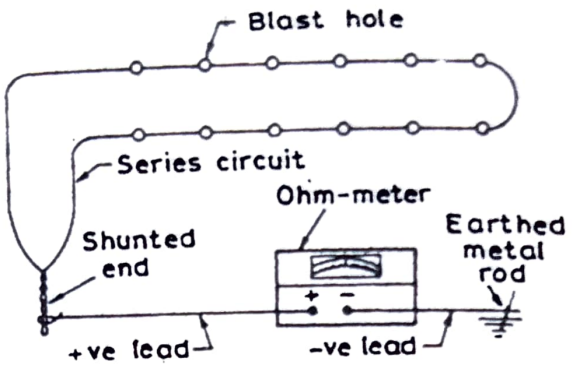


Fig. 19 Checking of Blasting Circuit by A Ohm-Meter

needle shall move and note down the resistance of the leadline. If the needle does not move there shall definitely be a break in the cable.

- For entire circuit : Connect the ends of the detonator's wires with the terminals of the Ohmmeter and note down the readings. It must match the calculated resistance.

The following inference may be obtained due to faulty system :

	Reading	**Inference**
(a)	Very low resistance compare to the calcutated resistance	Short circuit. Check the shorted leg wires for each detonator.
(b)	Very high resistance compare to the calculated resistance	Open circuit. Check the breakage of the circuit, poor connection, miss connection, etc.

Question 48 : What types of systems are available under the Shock Tube Initiation System? Describe 'NONEL'. What advantages may be offered by the Nonel or Shock tube system?

Answer : There are two systms available:

a) Nonel b) Raydet

Nonel termed for non-electric detonator. It consists of a flexible plastic tube having 3mm and 1.5 mm external and internal diameter respectively, inside of which is coated with an explosive composition having a detonating velocity of 1800 m/s. It is available in precut lengths. In one end of the tube is fitted with a non-electric delay detonator which is crimped to it while the other end is sealed. The end having detonator is lowered down into the hole through the explosive charge while sealed

end projecting outside the hole. The sealed end is initiated by a detonator or detonating cord. Nonel without a cap can not detonate on its own any commercial product. The shock wave passing through the tube is very mild and cannot even damage the walls of the tube as the quantity of explosive in the tube is very low. Nonel or shock tube system has two good characteristics compared to the detonating cord which are mentioned below :

a) Noiseless characteristics

b) It has no effect on the explosive column. Nonel can nicely be used in external disturbances like electrical storms, static electricity, stray currents, etc. and since each compound in the system is sealed in the factory it can be nicely used in underwater conditions.

Raydet developed by an Indian Company named IDL Chemical Ltd., is similar to Nonel detonating system.

Question 49 : Discuss 'RAYDET' a non-electric delay initiating device.

Answer : Raydet is a non-electric initiating device which combines the versatality, advantages of Electric Delay Detonators, Detonating cord and also overcomes some of their limitations. Raydet can be used for surface Blasting, Quarrying, Trenching, Large hole stoping, Ring hole blasting, Tunnelling, and other underground blasting operations.

Raydet (fig. 18) consists of a plastic tube with a very small quantity of reactive material in it's annular space. A delay detonator is factory assembled at one end of the tube. The other end is sealed to prevent ingress of moisture and other foreign matter. Upon initiation, a low order shock wave travels through the tube and reliably initiates the detonator. 20 Delay periods in the millisecond range and 12 periods in the half second range are currently available. Raydets are available with different tube lengths to suit customer's requirements.

Initiation of Raydets is carried out with detonating cord. Plastic connectors specially designed for hook-up of Raydet tube onto detonating cord are provided with every Raydet. Bunch blasting of upto 10 'Raydet Tubes' using a wrap of 'Detonating cord' is also possible especially in tunnel blasting, ring blasting, stoping etc., which makes hook-up very easy and less time consuming.

Advantages of Raydets

1. Raydet being a non-electric system, is immune to extraneous

electricity sources such as stray currents, static electricity and radio frequency energies emanating from walkie-talkies.

2. Being non-electric, it eliminates totally the risk of misfires of electric detonators due to earth leakage in highly conductive ore-bodies.
3. Raydet tube is insensitive to accidental initiation due to fire, impact and friction normally encountered during handling/ usage. Detonators used are conventional ones and are required to be handled like other detonators.
4. Raydet provides accurate down-the-hole delay and in surface blasting, provides true bottom priming. Bottom hole initiation is a pre-requisite for efficient bench blasting, results and reduces the risk of Flyrock generation, formation of toe and airblast noise.
5. Upon detonation, 'Detonating cord' generates a lot of shock and gas. In boreholes, this results in ejection of stemming material resulting in premature venting of detonation products and also partial de-sensitisation of non-cap-sensitive explosives in the borehole. Detonating cords of higher grammage can also cause, at times, detonation of explosives across the borehole diameter (instead of along the column). The explosive thus initiated gets consumed before steady state VOD is reached and thus results in less energy release.

   In view of the small quantity of reactive material present, the Raydet tube is practically non-disruptive. It does not cause any disturbance to the stemming and also ensures full energy release from the explosive present in the borehole. Raydet tube would not initiate any cap-sensitive explosive in the borehole unlike other delay systems based on low energy detonating cord (LEDC).
6. Where needed, Raydet down-the-hole in conjunction with surface delays, provides a large number of sequential delays. Multiple delays enhances fragmentation, well spread out muckpile and minimises ground vibrations through reduced maximum charge per delay.
7. Initiation of explosive decks on different delays in the same borehole is possible, such as for controlling ground vibrations

in surface blasting and in large hole stoping in the underground.

8. 'Bunch Blasting' of upto 10 Raydet tubes with a single wrap of detonating cord is possible in tunnelling and stoping operations, which makes connections easy, less cumbersome and time saving. In bunch blasting, use of plastic connector is eliminated.
9. Raydet does not cause any desensitisation of booster-sensitive explosive in the blasthole. Hence full energy of the explosive can be realised in the blasthole.
10. It does not cause any cratering effects from the collar portion of the blasthole.
11. Explosive decks in the same blasthole can be initiated on different delay intervals to reduce maximum charge per delay.
12. Since the detonation wave is contained within the tube, it significantly reduces airblast noise when used on the surface in place of detonating cord trunklines.

Raydet initiation system resulted in considerable improvement in production, productivity, cost reduction, better fragmentation, higher loading machine output, and presented itself as a better environmental friendly system.

Raydet Products and Uses

- Raydet HS (Half second) Delay no. 1-12. Uses : Tunneling, Shaft sinking, etc.
- Raydet ms (Milli Second) Delay No. 1-20,

  Uses : Tunneling, shaft sinking, underground stoping, ring hole blasting, long hole stoping, drop raising, blasting in opencast mines, trenching, etc.
- Raydet DTH (Down the hole) 475 and 500 ms.

  Uses : Long hole stoping, drop raising, vertical crater retreat (VCR) blasting, pipeline trenching, etc.
- Raydet TLD (Trunk line delay), 0, 17, 15, 42 and 67 ms

  Uses : In open cast mines and trench blasting as surface delay.
- TLD 'o' ms delay can be used for 'Bunch Blasting' of Raydets (ms/HS) in tunneling/excavation for hydroelectric, rail and

road projects, shaft sinking and also in the underground metalliferous mines.

Delay periods available

Various Delay periods available are shown in the table-20 given below.

Raydet delays are factory assembled and are available with tube lengths of 3, 5, 9, 10, 12, 15, 20, 22, 25, 30, 35, 40 and 45 metres. Tags indicate the delay number and tape fastening the tube in a coil indicates the length of tube. The tags are colour coded viz Red for millisecond and Blue for Half second. Raydets are packed in paper covers which in turn are placed in a wooden case. Upto 100 numbers of Raydets are packed in a case.

Table 20

Delay Number	Short Delay (Milli seconds)	Long Delay (Half seconds)
1	50	0.5
2	75	1.0
3	100	1.5
4	125	2.0
5	150	2.5
6	175	3.0
7	225	3.5
8	275	4.0
9	325	4.5
10	375	5.0
11	425	5.5
12	475	6.0
13	550	
14	625	
15	700	
16	775	
17	850	
18	925	
19	1000	
20	1100	

Question 50 : What problems may be associated with the conventional non-electric blast initiation system? What is Redundant system? Explain.

Answer :

The potential problem with convertional non-electric blast initiating system (surface delay, cord downline) are as follows :

- Cutting off the path will result in non-blasting of the adjacent holes.
- Results development of back cracks of overburden.
- Exposes undetonated explosives.
- Loss of production and productivity is more.
- Problems and hazards arise to blast the remaining holes in next attempt.
- Severing of cord downline by hole shifting prior to initiation.

Redundant System

A rdundant system is a non-electric, blast initiation system which employs multiple single paths to blast holes in order to accomplish reliable hole firing and avoid surface delay shut downs. A redundant trunk line delay (RTD) consists of a length of single tubing with delay detonators inside the block connectors (RTD blocks) at both ends. This permits firing from either end. Since redundant system provides a bi-directional system with two or more paths of initiation of each borehole, the system provides blast designers to maintain desired delay sequence even though cutting off a path will initiate the signal to follow another path to fire the hole.

Question 51 : What is LEDC system? Explain, what types of LEDC systems are available? What is Primadet and Detaline system? What advantages one can get with LEDC system?

Answer :

Low Energy Detonation Cord Delay System (LEDC System)

In this system two initiating devices are combined to form an integral system in such a manner that the detonation from one device to the low energy cord is attenuated to initiate the delay in the other device viz. the delay detonator. So here retaining all the advantages of the precise short delay blasting of electric circuits, the sequential firing mechanism is introduced into the non-electric circuit. The following LEDC systems are available :

i) Primadet system

ii) Anodet and Toedet system

iii) Detaline system

In Primadet system, a miniaturised (approximately 0.6 gm/m of PETN) detonating cord is used. In Detaline cord system 0.4 gm/m PETN are protected in thin textile fibre and covered by a seamless, tough, abrasion resistant outer plastic jacket. Anodet system uses 1.0 gm/m PETN cord. Detaline system requires special detaline starter and has possibility of in-the-hole delay system.

Advantages of the LEDC system can be summarised as follows :

i) It eliminates concern for all electrical hazards of premature initiation by stray or extraneous electrical currents normally encountered in mining operations including induced currents from nearby lightning strikes.

ii) It is immune to premature initiation from static electricity that has been generated during pneumatic loading operations or from radio frequency energy. It may be used without risk in close proximity to radio trasmitters, radar or similar installations.

iii) It is highly resistant to accidental detonation from rough handling during loading operations.

Since the system is immune to static charge generated during pneumatic loading, maximum air pressure can be used with the various ANFO pneumatic loaders.

Question 52 : What are the common types of exploders used in mines? Explain them briefly.

Answer : The common types of exploders which are used in the mines are as following :

a) **Magneto exploder or dynamo exploder :** Here the electrical energy is generated by means of dynamo or magneto. An armature revolves between the poles of a magnet either by gearing with a rotary handle or by a rack and pinion system and the value of the voltage produced depends upon the speed of the revolving armature and the flux created by the magnets. In otherwords this type blasting machine has a amall generator inside which is activated. When the rake bar is pushed down, the generator produces a direct current which fires the electric

detonators. The harder and quicker the stroke in case of bar type exploder the more current will be generated by the generator. In case of twist generator type exploder harder and faster is the twist, higher current will be produced. A high and low tension exploder produces a voltage of 125 volts and 15 volts respectively. Example : Little Demon, Schaffler–750 (suitable to fire 100 shots a time) exploder etc.

b) **Battery operated condenser discharge type :** In this type of exploder first of all a capacitor is charged either by battery or by a dynamo contained in the exploder and the same is discharged through the shotfiring circuit during blasting.

The capacitor or bank of capacitors are charged by pressing the 'Charge' button or switch. When the signal light glows, the machine is charged and ready to fire. Firing can be done by pressing the 'fire' button along with pressing down the 'charge' button. Fig. 23 shows the an arrangement of the blasting machine.

Here current is generated from a battery of 4 to 12 volts dry primary cells (having 1.5 volts per cell) connected in series. It is operated by a detachable key. Due to the low voltage and the absence of the inductance it can not produce any dangerous spark which can ignite fire damp.

This battery operated condenser discharge type exploder is designed to fire upto six electric detonators with copper lead wires, connected in series. The electric detonators available in India have tinned steel lend wires and hence have a higher electrical resistance (2.5 - 4.1 ohms). To ensure that the circuit resistance is within 25 ohms, not more than 4 such detonators should be connected in series. Safe limit of resistance is clearly indicated by the end of the white zone and the beginning of the red zone on the indicator scale of the exploder.

The M.E. 6 Exploder (which is capable to blast 6 shots at a time) is approved for use in all categories of coal seams. It requires a 67.5V battery for charging the condenser and a 1.5V battery for energising the moving coil testing indicator. A 67.5V radio battery type 468 manufactured by Union Carbide in India, is approved for use, but it must be replaced after 300 operations, or after 3 months from the date of manufacture whichever is earlier.

M.E. 12 and M.E. 30/4 (25 shot) exploders are approved for use in all coal seams.

Schaffler - 350 exploder and Rhino - 3, Rhino - 12 and Rhino - 25 exploders are permitted to be used only in degree-I and degree-II gassy coal mines; whereas Schaffler – 750 is suitable for firing 100 shots in opencast coal mines.

Question 53 : What is Blaster Galvanomater, Blaster Ohmmeter and Blaster's Multimeter?

Answer : *Blaster Galvanometer :* It is an apparatus to measure the resistance of the broken wire, short circuit and to assess the compatibility of firing shot by an exploder.

Blaster Ohmmeter : It is also used to measure the resistance of the blasting circuit and the detonator but with more accuracy and at a higher range.

Blaster's Multimeter : This apparatus measures all the electrical parameters like current, voltage (both A.C. and D.C. voltage and milli-volts) and resistance. With this instrument a blaster can measure the resistance of individual and total blasting circuits, test the blasting area for extra currents, measures the leakage current between a detonator and the ground, check the continuity of the circuits, short circuits, connecting firing lines, and also it tests individual detonators.

Question 54 : What are the various advantages and disadvantages of shot firing with series, parallel and parallel-series circuits?

Answer :

Series Circuit

In this system the firing current should be atleast 1.5 amperes for proper functioning of the electric detonator with the D.C. circuit, and the continuity of the circuit can be tested accurately. The circuit necessitates low current at high voltage and they are not generally employed with mains firing. The required voltage in this system depends upon the number of shots to be fired (shots are generally fired not more than 30 in numbers), resistance of the detonators, lead wires, shot firing cables, etc. In this system the shots can be counnected very easily (Fig. 20).

Parallel Circuit

In this system every electric detonators are provided with an alternative path for the current and hence the continuity of the complete circuit cannot be tested. The circuit necessitates a minimum current (A.C.

or D.C.) of 1 ampere per. detonator. This type of circuit is generally used with mains firing. Since in this case the first shot gets more current than the last one hence delay interval should not be less than 50 ms in case of short delay detonator and also the largest delay detonator should be connected nearest to the firing station. In this system more manpower can be employed to charge the holes and to connect the shot holes to complete the circuit (Fig. 21).

Parallel-Series Circuit

This type of circuit is generally used where more than 30 number of electric delay detonators are to be fired and where the series resistance is very high. Here the number of electric detonators connected in series is limited by the voltage and number of series circuit connected in parallel is limited by the amount of current flow. Generally each of the series circuit contains the same number of electric detonator and hence they all have more or less equal amount of resistance (Fig. 22).

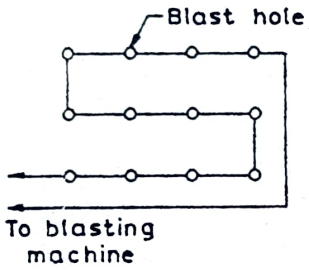


Fig. 20 Series Circuit

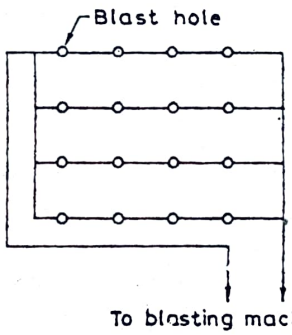


Fig. 21 Parallel Circuit

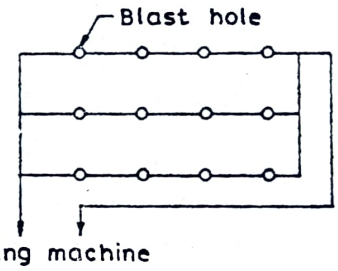


Fig. 22 Series-Parallel Circuit

Calculation

For Series circuit :

Let the number of electric delay detonators are connected in series is 30 each of which has a resistance of 2 ohms and let the resistance of the connecting wire and the blasting cable is 10 ohms and 2 ohms respectively,

So resistance of 30 detonators 30 X 2 = 60 ohms
Resistance of the connecting wire – 8 ohms
Resistance of the blasting cable – 2 ohms
Total Resistance – 70 ohms

So the approximate amount of voltage required = I X R = 1.5 X 70 =105 volts (Actually minimum requirement of current per series is 1.2

amperes, although, for calculation purpose it is considered on higher side say 1.5 amperes).

For Parallel Circuit :

Resistance of 30 electric detonators –	0.066 ohms
Resistance of the connecting wire –	8.000 ohms
Resistance of the blasting cable –	2.000 ohms
Total Resistance –	**10.066 ohms**

Minimum requirement of current for this case is 1 ampere per detonator. So total current. = 30 X 1 = 30 amperes.

Hence voltage requirement = 30 X 10.066 = 301.98 volts.

Parallel-Series Circuit :

Let there be 30 electric detonators distributed in this manner - In each series connection number of electric detonators is 10.

So number of series circuit connected in parallel connection is 3.

So resistance of 30 electric detonators –	6.66 ohms
Resistace of the connecting wire –	8.00 ohms
Resistance of the blasting cable –	2.00 ohms
Total Resistance –	**16.66 ohms**

So the requirement of current = 1.5 X 3 = 4.5 amperes

Hence voltage required = 4.5 X 16.66 = 74.97 volts.

Question 55 : In a development face 10 holes are drilled and charged with AJEX-6 explosives along with electric delay detonators connected in series. The each connecting wires length is 1 m. The lead line is 100 m in length. Calculate the total resistance of the circuit.

The resistance of each detonator	:	1.4 ohm
The resistance of connecting wire	:	0.03 ohm/m
The resistance of lead line (consisting of 2 wires)	:	0.008 ohm/m/wire

Find out the total resistance of the system.

Answer :

Total length of the connecting wire (2 wire/detonator)	:	10 x 2 = 20 m
Therefore resistance Rc.	:	20 x 0.03 = 0.6 ohm
Total resistance of detonators Rd	:	1.4 x 10 = 14 ohm

The resistance of the lead line Re : 2 x 100 x 0.008
= 1.6 ohm

Hence the total resistance of the system

Rt = Rc + Rd + Re = 0.6 + 14 + 1.6 = 16.2 ohm.

Question 55 A : In the above mentioned example if there are 2 series circuit each consisting of 5 detonators and thereafter if they are connected in parallel, what will be the total resistance of the system?

Answer : Total length of connecting wire in each series circuit :

5 x 2 = 10 m

Therefore Rc_1 = 10 x 0.03 = 0.3 ohm

Resistance of 5 detonators Rd_1 = 5 x 1.4 = 7 ohm

The resistance of each series circuit

$R_1 = Rc_1 + Rd_1 = 0.3 + 7 = 7.3$ ohm

and R_2 = 7.3 ohm

Therefore $1/R = 1/R_1 + 1/R_2 = 1/7.3 + 1/7.3 = 2/7.3$

or R = 7.3/2 = 3.65 ohm

Hence total resistance of the system = R + Re = 3.65 + 1.6 = 5.25 ohm

Question 56 : Discuss briefly about the Mains Firing System.

Answer : When large number of shots are required to be fired, this system is used particularly in the systems of parallel or parallel- series firing circuits. In this system, firing of the shots are done from the A.C. mains with a voltage in between 220 to 660 volts and each shot is provided with 3 to 4 times the direct current. Use of higher voltage is not adviseable since it may cause electric shock. Since with this system many safety features against lightning, stray currents, induced currents, safeguard against wrong operation of firing switch are involved, the system is very costly both for installation and maintenance. During firing, blasting cable is only used in this system which is separated from other cables, water or C.A. pipe lines, ventilation ducts, etc. and carried on insulation. The system consists of 3 main units :

Unit-I consists of grounded metal box containing the main switch, fuses and receptacle for a plug in connection between the unit-1 and unit-2.

Unit-II : It is made of wooden box and is connected with the Unit- I with the help of a cable. It contains a firing switch and a pilot light to

indicate the presence of the firing current. It is operated manually to 'on' the circuit. The door of the unit is made in such a manner that it will close only when the firing handle is returned to the 'off' position with the firing lines shorted.

Unit-III : This unit is situated in between the face and the unit- II to ensure that the blasting lines are disconnected from the cable and properly sorted. The door of the unit will close only when the short circuiting switch of the unit returns to the shorted position.

Question 57 : Discuss various firing system in the open cast mines.

Answer :

Firing System

It may be basically divided into 4 types which are as follows :

1. Electric Initiation : This system comprises of electrical detonator connected to the blasting cable through legwires and the blasting cable ultimately is connected to an exploder which provides the necessary electrical current to fire the shots. This system is applicable for multiple charge firing, in any atmospheric conditions, quites complex in design and usage, may cause a hazard under stray currents but the system can be checked by a multimeter. In opencast firing with electric detonator the length of the leg wires are very large (it is equal to length of the borehole + 0.5 x spacing interval + some allowance). The delay range in case of short delay period is 25 ms whereas in case of longer delay period 50, 75, 100 ms. The initiation of the blastholes may also be done by the sequential timer.

2. Detonating Cord : The charge can be initiated by the detonating cord. The cord can be initiated in 3 ways ;

(a) with a non-electric delay detonator

(b) with a ms electric delay detonator

(c) with a ms connector

In case of non-electric delay detonator to be used for the individual hole attach a non-electric delay detonator at the detonating cord downline at the collar of the hole whereas in case of delay a row or group of holes, attach the detonator to the trunkline between the holes.

In case of electric delay detonator to be used in the surface delay

(Note : ms is milli-second)

attach detonator either to the downline of individual holes or in the trunklines connecting rows of group of holes.

In case of ms connectors to be used, the latter may be jointed into the trunkline between boreholes or group of boreholes.

3. By Non-electric delay detonators.

The NONEL systems used for blasting in the surface mine are as follows :

By Nonel Shocktube - already discussed

- Noiseless lead-in
- Noiseless trunkline delays
- Detonating cords - already discussed.

Noiseless trunkline delays are the combination of trunkline and surface delays having a wide range of delay periods.

Noiseless lead-in is a shock tube leadline that can be used with the NONEL or with a detonating cord.

4. By Hercudet firing system :

In this case a gaseous mixture is transmitted to the boreholes by plastic tubes and connections. A gas supply unit introduces the gas to the tubing system and a Hercudet blasting machine detonates the gas.

The system comprises of Hercudet detonator, a duplex tubing, a single tube trunkline, plastic connectors, Hercudet blasting machine and other accessories. The Hercudet blasting machine controls the amount of fuel and oxidizer to be needed in the firing circuit. After the system has been charged, the blasting machine control lever is actuated to the ARM position and fire button is pushed. A low energy gas detonating impulse travels through the firing network of the plastic tubing at a rate of 2400 m/s to ignite the fuse element in the Hercudet blasting detonators. The blasting accessories comprise of gas supply unit (contains 3 small bottles, one each of O_2, N_2 and fuel for supplying the gaseous mixture to fire the Hercudet system), Hercudet tester, Hercudet pressure test module, etc.

Question 58 : What is sequential blasting? How can it be achieved? What is sequential blasting machine?

Answer : The holes which are charged, if blasted in sequence is called the sequential blasting. It can be achieved in the following ways :

1) By using ordinary electric delay detonator, long electric delay

detonator, Milli second electric delay detonator, Electronic delay detonator, Magnadet delay detonator, etc.

2) By using Raydet, Primadet, Anodet, Toedet, Detaline system and also by using non-electric milli-second delay detonator or electric detonator with the detonating cord.

3) In blasting with the safety fuses, different lengths of the safety fuse are cut from each charged hole and then lighted in the desired order or by connecting different cut lengths of the safety fuse from each charged hole and connecting them with a length of igniter cord in the desired order or by connecting standard fuse length with the igniter cord in the desired order.

4) But most simple way it can be achieved by blasting one hole at a time either with the safety fuse or by special ordinary detonator attached to the safety fuse. For more details consult question Nos. 78, 79, 80, 120, 121, etc.

Sequential Blasting Machine

The sequential blasting machine is a capacitor type blasting machine capable or energizing multiple electric detonator circuits in a programmed time sequence.

A sequential blasting machine fires sequentially large blasts with better fragmentation and less vibration. With this machine full production blasting of large number of holes is possible which otherwise would be curtailed if a regular single circuit machine is deployed. This sequential blasting machine for firing multiple row of delay electric blasting caps can be set to fire one after another sequentially at a preselected time interval from 1 to 999 milliseconds.

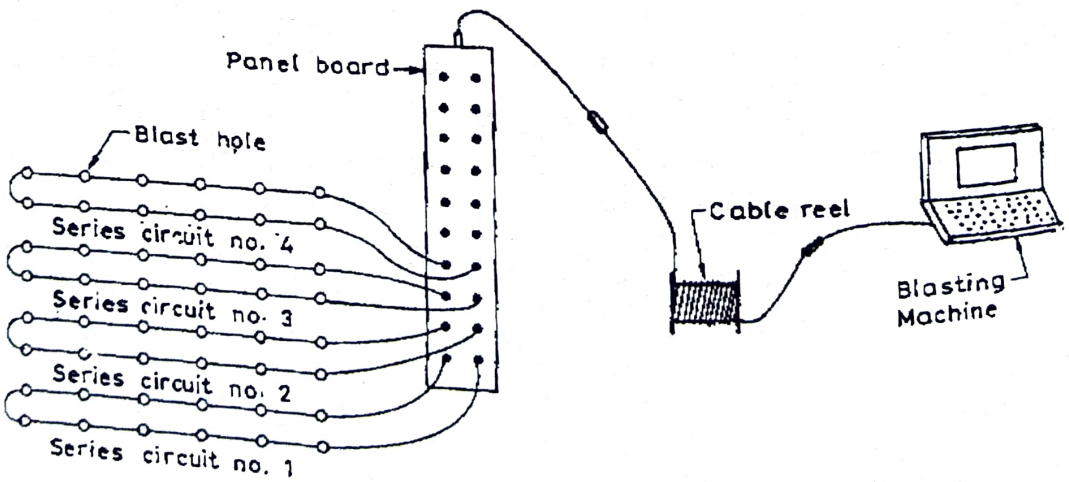


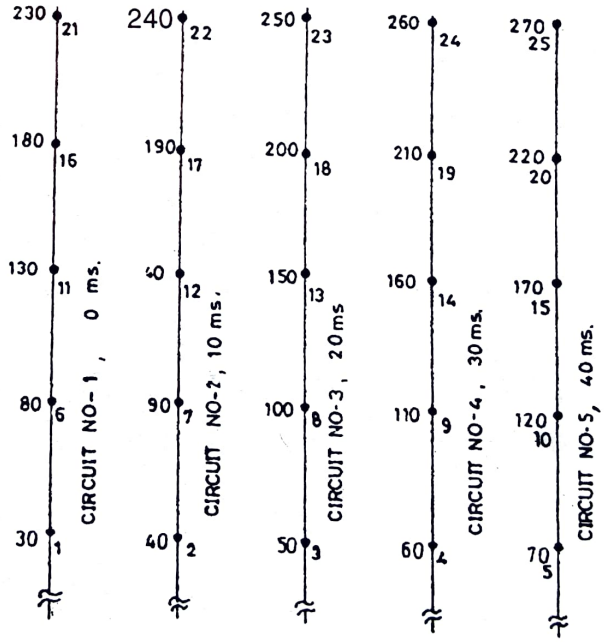
Fig. 23 Arrangement of Blasting Machine

The machine is condenser discharge type having sequential timer that assists the detonation of a number of detonators at precise but different time intervals. The machine has number of different firing circuits which are programmed to fire one after another at a predetermined time intervals and permits detonation of much larger blast per hole.

The sequential timer could be considered multiple blasting machine (10 or more machines) in one unit because with it a blaster can detonate a number of shots in a timed sequence. When this machine is used along with the delay detonator, the timer provides a part of the delay and the detonator provides the rest part of the delay. The circuits attached with the timer arranged in such a manner that the series of holes nearest to the free face to be initiated first, the next nearest series to be fired thereafter and so on. For clearly understanding consult fig. 23(a).

The sequential timer allows the blaster to double or tripple the size of the shot and improves fragmentation and control flyrock without enhancing noise and vibration.

Since the sequential blasting machine does not transmit the impulse until the delay has passed, there may be chances of cutting off the connecting wires due to the movement of the rock. So blasting circuit wiring should be carefully done and there must be some slackness in the connecting wires for allowing the movement of rockmass. This type of blasting machine is used for huge blasting in surface mines and tunneling works. In the Fig. 23(a) it has been shown that there



1, 2, 3.....etc. are the hole numbers to be blasted in order. 30, 40, 50.....etc. are the delay timing.

Fig. 23 (a)

are 5 rows in a bench, consisting of 25 holes and 25 delays, minimum time between delays are 10 ms and hence blasting machine is set on 10 ms.

Question 59 : What is the maximum number of shots a blaster can fire in a shift?

Answer : As per coal mine regulation 1957, (the sub-regulation no. 166(5)) the maximum number of shots that a shot firer can fire in one shift is

i) If the seam is of degree II and degree III gassy and if

a) a single shot exploder is used — 40 nos.

b) a multishot exploder is used — 80 nos.

ii) In other seam if

a) a single shot exploder is used — 50 nos.

b) a multishot exploder is used — 100 nos.

iii) In opencast mines if

a) a single shot exploder is used — 60 nos.

b) a multishot exploder is used — 120 nos.

B. As per MMR 1961, (the regulation No. 160(5)) the maximum number of shots that a blaster can fire in a shift is :

a) if blasted electrically or by means of an igniter cord — 80 nos.

b) in other cases — 50 nos.

However where special conditions exist, the Regional Inspector may by an order in writing and subject to such conditions as he may specify therein, permit a large maximum to be fixed. But where the Regional Inspector is of the opinion that the observance of the regulations are not being properly made, he may also reduce the number of shot to be fired by a blaster in a shift.

Question 60 : Discuss briefly the following :

a) cables for single shot firing, (b) cables for multishot firing, (c) cables for shotfiring in shaft.

Answer :

For single shotfiring

Twin core cable, each core consisting of a conductor of at least 4 copper wires of not less than 0.45 mm in diameter, with insulation and further protection against mechanical wear and damage not less than that

of polyvinyl chloride 0.625 mm in thickness wiih a figure of eight construction or of vulcanised rubber of P.V.C. 0.5 mm in thickness. The two cores twisted wormed and covered with a sheath of cotton braid, rough rubber or P.V.C. should be used.

For Multi-shotfiring

Single core cable, consisting of conductor of at least 3 copper wires of not less than 0.9 mm in diameter, insulated to withstand at least 250 volts, and further protected against mechanical wear and damage should be used.

For Shotfiring in Shafts

Two core cables, each core consisting of a conductor of at least seven copper wires of not less than 0.725 mm in diameter. insulated to withstand at least 250 volts and adequately protected against mechanical wear and damage should be used. Cables should be at least 300 m long.

Queslion 61 : What are the tools to be provided to a shot firer? Discuss about the break detector.

Answer :

Shotfiring tools

A shotfirer should be provided with and should make use of the following tools :

- A suitable electric lamp and a stop watch.
- Wooden stemming rod.
- Brass or wooden scraper to clean the holes.
- A knife to cut fuses unless machine crimped fuses are available in safety fuse firing (where permissible).
- A pricker made of wood or some non-ferrous metal.
- A suitable break detector preferably of the following specifications :

(i) A straight stiff rod or tube made up of copper or wood with a flat prong at one end which forms an integral part of it, or is securely attached to it. The prong is made of or tipped with hard wearing metal.

(ii) Length of rod or tube : It should be 15 cm longer than the

depth of shothole in which the detonator is to be used but in no case less than 1.36 m. Diameter of rod or tube shall be half the diameter of the shothole but in no case more than 1.27 cm; length of prong shall be not less than 2 cm and more than 2.5 cm from the centre of the rod or tube to the tip of the prong. Width and thickness of prong at the tip shall be more than 2.4 mm. The break detector may incorporate a scraper attached to the outer end. The scraper attachment should comprise not more than half a disc.

Question 62 : Write short notes on the following :

a) Deflagration

b) Water or gel-filled stemming

Answer :

(a) Deflagration

If the reaction of decomposition of an explosive moves through the explosive charge slower than the velocity of the sound it is termed as deflagration. The type of explosion from the gun-powder and other low explosives follow a rapid form of combustion in which the particles burn at their surfaces and expose more and more of the charge until it is totally consumed. The deflagration of low explosives yield more gases slowly at lower but substantial pressure, thereby causes a heaving and pushing action in place of shattering or blowing action respectively. So low explosives, if firedamp conditions permit, are more applicable to the blasting of soft plastic rock and coal where big lumps are desirable.

(b) Water or gel filled stemming

Presently water stemming is slowly replacing the conventional clay and sand stemming worldwide for blasting in the underground coal mines. Water or gel filled ampoules have been used to seal the hole, quench/cool the flame of the explosion, suppress dust and also to absorb fumes. It is a very efficient stemming system, and reduces the explosive charge per tonne of coal production and hence very cost effective. The gel filled ampoule filled with water and organic substances and sealed under factory conditions, has flame-resistant non-abrasive cylindrical sheath. Water ampoules consisting flame resistant non-abrasive cylindrical sheath (PVC), around 0.35 to 0.4 m long, are generally self

sealing types and water may be filled on the surface or in the underground. Sometimes cylindrical P.V.C. bags of 30 mm diameter (approx) filled with water after making a knot at the empty end are also used as stemming material. During charging a shothole first conventional stemming material made of clay and sand is inserted into the hole, then explosive cartridge and followed by one or two water or gell-filled amoules or water filled P.V.C. bags are inserted. Finally the mouth of the holes are stemmed with the conventional sand-clay stemming at a minimum length of 0.35 to 0.4 m. With the use of water or gel-filled stemming risk of coal dust/fire damp explosion is reduced, blasting efficiency is improved due to the cushioning effect beside other advantages as already mentioned.

Out of the many types of water ampoules are available in the market some of them are mentioned in the following paragraphs–

Normal type ampoules–They are manufactured from PVC material and are generally 375 mm long and 50 mm in dia. The thickness of PVC wall is around 0.15 mm and the ampoules consist of a self-sealing valves. Ampoules are filled with water near to the underground blasting faces and two ampoules should be used per hole when the length of holes exceeds 1.3 mm. One ampoule shall be inserted first, then the chargc then anothcr ampoules, and finally the hole shall be stemmed with ordinary stemming material.

Ampoules filled with water in the shotholes–This type of ampoules are filled with water in the shothole through the pistol-grip filling valve by a filler tube. After inserting the explosive cartridges into the shot holes empty ampoule attachcd with filler tube, is pushed inside the hole upto the charge with the help of wooden stemmer and water at a pressure around 1 kgf/cm^2 is supplied in it. When the relief valve of the ampoule opens, it will indicate that ampoule is totally filled with water and the filling tube is withdrawn from the hole keeping the same operating water pressure at constant. This typc of ampoules are 450 mm long and 50 mm in dia. having wall thickness (of the ampoule bag) around 0.2 mm.

Re-entrant type ampoules–They are longer in size having inside pocket for housing explosive cartridges separating from water pocket. by the ampoule skin. Sizes of ampoules are generallY 1.05m in long having a wall thickness around 0.2 mm. Tlas type of ampoules provide

better protection against the danger from methane explosion.

Gel Ampoules–This type of ampoules are made from semirigid polythene tube, length and diameter of an ampoules are generally 41.25 cm and 37.5 mm respectively. The ampoules are filled under pressure with gelatinous content (96% of water mixed with chemical gelling and bactericide agents) and pre splitted in 4 places on the ampoule 100 to 125 mm away from its in oye end. After the hole is charged with explosives the gel ampoules is inserted into the hole and stemmed with a wooden stemmer. Due to the stemming pressure same amount of gel will came out from the ampoules and the hole will completely be filled up. It will be better if ordinary stemming material is used for stemming after the gel ampoule is completely inserted into the hole.

Question 63 : What is a booster? Give example.

Answer : Normally a high detonation velocity explosive enclosed in a small metal case is necessary to detonate a few kinds of slurry explosives, ANFO mixture, etc., which is termed as booster (cast booster). The booster does not contain an initiating device and used for cap insensitive explosives. The boosters are designed to give a high detonation pressure. They have a high inherent strength, density, and high water resistant properties. The boosters are not readily detonated by shoc, friction or heat. However a booster may be a conventional high explosive in which case one of its cartridges is to be primed with a detonating fuse. Example : Toe blast, KELVEX-800 etc.

Example of Cast Booster

i) Primex booster. manufactured by IEL : It is used for non-cap sensitive explosive and is water resistant. It is initiated by the cordtex detonating fuse and detonate at about 7000 m/s. Sizes available are of the range of 23, 37, and 50 mm dia, with weights of 20, 100 and 250 gms respectively.

ii) Avimex cast booster manufactured by IGEL has a high VOD, high density and high strength cast booster and contains a mixture of PETN and TNT for initiating and bosting non-cap sensitive explosives such as NCN-400, NCN-500, NCN-600 including ANFO and Emulsion Explosives. It is available in 100 gms, 250 gms and 500 gms sizes.

Density : 1.6 + 0.5 gms/cc, VOD : 7000 + 500 m/s.

Question 64 : What is a primer and booster? Give example.

Answer : These are the explosives/containers of explosives into which a detonator or detonating cord is inserted for detonation of the main charge.

A primer or booster consists of the combination of a strong explosive cartridge and a detonator or only a detonating cord. The primer is placed at one end of the column of charge pointing the detonator towards the other charge in such a manner that by initiation of it, the remaining column of charge will set off. Similarly when detonating cord is to be used it should be attached to the first cartridges to be loaded into the hole so that it will initiate other cartridge present in the borehole.

Example :

Aquadyne, Opencast Gelignite (OCG), Pentolite booster, etc.

For more details consult question no. 66.

Question 65 : What are the basic tests which are carried out to determine the characteristics of an explosive?

Answer : The following tests are carried out :

- Ballistic mortar tests, underwater test or Trauzi lead block test for the determination of weight strength of the explosive.
- Dautriche Test for knowing the velocity of detonation.
- Heat test to determine the chemical stability of the explosive. It is generally done by the Abel heat test.
- Fall hammer test for knowing the impact sensitivity.
- Ardeer Double Cartridge test for determining the sensitivity of an explosive to the propagation of detonation along the maximum gap between the cartridges.

Question 66 : Give specifications of some of the primer/booster for large hole blasting in opencast mines.

Answer : Specifications of primer/booster for large hole blasting in opencast mines are given in Table 21.

Question 67 : Enumerate various characteristics of bulk explosive system in use in India.

Answer : Various Characteristics of bulk explosive system in use in India is given in Table 22.

Table 21

S. No.	Manufacturer	Type	Detonation Velocity (m/sec)	Density (g/cc)	Sensitivity	Classification	Cartridge diameter (mm)	Cartridge weight (kg)	Cartridge length (mm)
1.	IDL Chemicals Ltd.	AQUADYNE	3400-4000	1.10-1.20	Sensitivity to No. 6 and No. 8 detonator & detonating fuse	Class-2	75 83 125 175 200 and above	2.08 2.76 6.25 8.33 12.50	
	- do -	TOEBLAST	4000-4500	1.2-1.25	Sensitivity to No. 6 detonater & detonating fuse	Class - 2	- do -	- do -	
2.	Karnataka Explosives Ltd.	KELVEX-800	4500-4800	1.20-1.25	Sensitivity to No. 6 detonater & detonating cord	Class - 2	83 125 200	2.78 6.25 12.50	460-480 470-480 410-420
		KELVEX-600	3800-4200	1.18-1.22	- do -	- do -	- do -	- do -	- do -
3.	Maharashtra Explosives Ltd.	MEXBOOST	4800±200	1.25±0.03	No. 6 detonator	Class - 2 Explosive	83 125 175 200	2.78 6.25 8.33 12.50	
	- do -	MEXPRIME (Booster)	4200±200	1.18±0.03	- do -	- do -	- do -	- do -	
	- do -	MEXPRIME-XX (Booster)	4000±200	1.12±0.02	- do -	- do -	- do -	- do -	

Table 21 (continued)

S. No.	Manufacturer	Type	Detonation Velocity (m/sec)	Density (g/cc)	Sensitivity	Classification	Cartridge diameter (mm)	Cartridge weight (kg)	Cartridge length (mm)
4.	Indo Gulf Explosives Limited	IGEL PRIME (Excellent water resistance capacity, Weight strength-68% of blasting gelatine)	3900±200	1.15-1.25	No. 6 detonator, detonating cord	-	125 175 200	6.25 8.33 12.50	
	- do -	AVIPRIME (Excellent water resistance capacity, Weight strength - 76% of blasting gelatine)	4200±200	1.25-1.32	- do -	-	83 125 175 200	2.78 6.25 8.33 12.50	
	- do -	AVIBOOST (Excellent water resistance capacity, Weight strength - 90% of blasting gelatine)	4500±200	1.25-1.28	- do -	-	- do -	- do -	

Table 22

CHARACTERISTICS OF DIFFERENT BULK EXPLOSIVE SYSTEMS IN USE IN INDIA

1. IBP Co. Ltd. Site mised sluries :						
Indogel	614	634	654	674	704	734
Al %	-	1.5	3.1	4.8	9.0	10.1
Weight Strength (ANFO=1)	0.76	0.82	0.89	0.97	1.07	1.18
Bulk Strength (ANFO=1)	1.02	1.12	1.23	1.36	1.53	1.73
Density (gm/cc)	1.10	1.12	1.13	1.15	1.17	1.20
N (Moles/kg)	45.11	44.08	42.97	41.79	40.14	38.12
Site mixed emulsions :						
Indogel	1116	1136	1156	1176	1206	1236
Weight Strength (ANFO=1)	0.76	0.83	0.90	0.97	1.11	1.22
Bulk Strength (ANFO=1)	1.16	1.27	1.37	1.48	1.69	1.86
Density (gm/cc)	1.25	1.25	1.25	1.25	1.25	1.25
N (Moles/kg)	45.33	44.22	43.11	42.00	39.79	37.94

	Site mixed slurries	Site mixed emulsions
VOD (m/sec)	4200+200	5000+200
Water resistance	Satisfactory	Excellent
Borehole sleep time	2 weeks	
Booster	Recommended 0.1% cast booster (Minimum 250 gms)	
Recommended depth	up to 35 meters	

2. ICI (India)	GMS 1B	GMS 2B	GMS 3B
	Aluminised Base Ch-I	Aluminised Base Ch-II	Aluminised Base Ch-III
Cup density	0.8-1.20	0.8-1.20	0.8-1.20
VOD (m/sec)	3400-3800	3500-4000	3600-4200
Water proofness	Excellent		

Table - 22 (continued)

3. KEL

SM emulsion

Truck capacity-10t

Rate of pumping = 6t/hrs.

Sensitizer=Perlite

	KECOLEM				
AI,%	0	2	3	4	5
Density	1.10-1.15	1.15-1.20	1.15-1.20	1.15-1.20	1.15-1.20
VOD	4000-4500				
Minm. diameter	100mm				
Bulk	110	126	133	140	145
Sitrength (ANFO=100)					

4. Indo Gulf Explosives IGEL-SM NCN bulk explosive system

	NCN-100	NCN-400	NCN-500	NCN-600
Density	1.0	1.1	1.2	1.25-1.30
VOD (unconfined) m/sec	3200	3500	3800	4500
Weight strength	100	120	140	150

Booster required=2kg/1 ton of SMS
Rate of pumping=250 kg/min

5. Maharastra Explosives Ltd.

Specifications	MEXAN-1	MEXBLAST
Detonation velocity	3500+200 m/sec	4000+200 m/sec
Density	1.13+0.03 gm/cc	1.25+0.03 gm/cc
Sensitivity	Booster sensitive	Booster sensitive
Water resistance	Excellent	Excellent
Minimum charge dia.	100 mm	100 mm

Table - 22 (continued)

Booster requirements :

Cast booster of 50 : 50 PETN/TNT would be suitable for priming above mentioned slurry explosives and the recommended percentage is around 0.2% which may be tapered off based on actual trial and field conditions.

6. IDL Chemicals Ltd.

-	Emulking (Aluminised)
Desity	1.15
VOD (m/sec)	5000-5500
Bulk strength	120-140
Detonation pressure	126-136 K bar
Sleeping time	7 to 10 days.
Cap. of truck-6 to 12t	
Rate of pumping : 100-200 kg/min.	

7. Anil Chemicals

Amerind Mackissic truck, capacity 8-10t (of emulsion/ANFO)

	AN%	Density	Available energy per cc	
Master Col	100	0.90	1.02	1.05
Master EM Col 100	75	1.02	1.10	1.28
Master EM Col 500	50	1.03	1.12	1.12
Master EM Col 1000	25	1.12	1.16	1.41
Master EM Blast	0	1.20	1.20	1.44
Master EM Blast Super	-	1.25	1.22	1.47

VOD : 3500-5000 m/sec
Pumping rate=150-200 kg/min.

Question 67 (A) : Prepare a table comparing the properties of commercial explosives.

Answer :

Table 22 (A)

Properties of Commercial Explosives

Properties	Nitro-glycerine based Explosives	ANFO	Slurry	Emulsion
Constituents	Nitro-glycerine, Ammonium Ammonium Nitrate, Sodium nitrate, Nitro cellulose, Sulfur, Carbonaceous fuel, Antacids, etc.	Prilled Ammonium Nitrate and Fuel oil	Ammonium Nitrate, Sodium, nitrate Fuels, Water, Cross linking agent, Flake Aluminium, Guar gum, etc.	Ammonium Nitrate, Barium Nitrate, Sodium nitrate, Fuel oil, Wax, Aluminium,Chemical gassing agent or micro bulloons.
Oxidiser	Ammonium nitrate, Sodium nitrate	Prilled Ammonium Nitrate	Solid/liquid (Nitrate Solution)	Liquid salt, (Nitrate solution)
Fuel	Carbonaceous fuels, Sulfur	Diesel oil	Solid/liquid (Aluminium Sensitizer)	Liquid (Fuel oil/wax)
Sensitizer	Nitro-glycerine	-	EGMM, MAN, Cross linking agent	Chemical Gassing agents or Air bubbles or Micro ballons
Physical Condition	Powder/Gelatinous	Powder	Liquid gel	Greasy
VOD (m/sec)	2500 - 6000	2000 - 4800	3800 - 4200	4500 - 5500
Impact sensitivity (Nm)	0.2 - 0.6	3.0 - 3.5	> 10.0	> 10.0
Density (gm/c.c)	1.2 - 1.6	0.8 - 0.9	1.05 - 1.25	1.15 - 1.3
Relative bulk strength (% BG)	73 - 79	51 - 55	53 - 65	56 - 71
Water resistance	Ecellent	Nil	Excellent	Excellent
Temperature resistance	-17.37	32.2	4.0	4.0
For frozen area	Good	Good	Poor	Poor
Fume Characteristics	Good	Very/Good	Excellent	Excellent

Question 68 : Name some of the explosives manufactured by some of the explosives companies in India.

Answer : Explosives manufactured by some of the companies in India are enumerated below :

1. Nababharat Explosives Company

Manufacturing non-permitted explosives like Nava Column, Nova base, Nova prime, Nova boost, Detonating fuse : Nova cord-II.

2. Rajasthan Explosives and Chemicals Ltd.

Manufacturing detonating fuse : R-Cord-II

3. IBP Co. Ltd.

Manufacturing permitted explosives like Indocoal-1 (P_1), Indocoal-3(P_3), Indocoal-5(P_5), Large dia non-permitted explosives like Indogel-210, Indogel-230, Indoprime (S), Indoboost (S), Small dia. non-permitted Indomite-60, Indomite-80, Cast booster.

4. Apex Explosives Private Ltd.

Manufacturing large dia. non-permitted explosives like Apex-70, Apex-80, Apex-Gel-80, Apex-Gel-90

Cast booster : Pentax

Detonating fuse : Apex detonating fuse

5. Nobel Explosives Ltd.

Manufacturing permitted explosives like Nobel coal-I(P1), Nobel Blast-I(P_1), Nobel coal-3(P_3), Nobel coal-5(P_5).

6. Eastern Explosives and Chemicals Ltd.

Manufacturing permitted explosives like Coaldyne-I(P_1), Coaldyne-3(P_3), Coaldyne-5(P_5), large dia. non-permitted explosives like Ecogel-1, Ecogel-2, Ecodyne, Ecoblast.

7. Macwin Explosives

Manufacturing explosives like NCS-1, NCS-2, MacWinex, Booster - Macpentolite, Dctonating fuses - Macond-1

8. Tamilnadu Industrial Explosives Ltd.

Manufacturing permitted explosives like Telperma-1(P_1), Telperma- 5(P_5), large dia. non-permitted - Telcol, Telgexld, Detonator – Copper electric detonators, Copper delay detonators, Detonating fuse - Telcord-II with 10 gm charge.

9. Premier Explosives Pvt. Ltd.

Manufacturing permitted explosives like Colex-l(P_1), Colex-3(P_3), Colex-5(P_5)

Large dia. non-permitted explosives like Dynex-l, Dynex-2, Dynex-3, Dynex-C

Detonators – Copper electric detonator, copper delay detonator

Detonating fuse – Safcord-10

Question 69 : Classify the bulk delivery system. After enumerating various advantages and disadvantages discuss the following :

a) Plant mixed slurry/emulsion system

b) Mix pump truck

c) Repumpable emulsion/slurry and the Gelmaster system.

What is a Silo Storage Facility?

Answer :

Bulk Delivery System

There are at present 4 type delivery systems mainly in use and these are :

- Plant mixed slurry/emulsion.
- Site mixed slurry/emulsion.
- Repumpable slurry/emulsion.
- ANFO/HANFO delivery system.

Plant Mixed Slurry/Emulsion System

In this system premixed slurry manufactured at a satellite plant located in close proximity to the mine site is transported in delivery trucks equipped with a pumping mechanism. The explosive prepared under controlled conditions and exactly tested for actual yield requirement is directly pumped into the borehole.

Advantages of this system are :

- Robust and reliable system.
- Explosive prepared under quality control supervision.

Disadvantages of PMS system are :

- Explosive is carried to the mine site.
- Large safety distance for plant.
- Delivery of a single product.

Mix Pump Truck

A mix pump truck carries ingredients in seperate containers housed on the truck and the actual mixing takes place at the site of blasting over the borehole. Calibrated quantities of ingredients are metered and blended together before being pumped to the borehole. The explosive attains the desired density and sensitivity after a short period of time within the borehole.

Advantages of this system are :

- Products with varying energy/density levels can be offered.
- Non-explosvie ingredients are carried at all times.

Disadvantages of mix pump truck system are :

- No checking is possible as the explosive attains explosive properties only after charging in the borehole. Further explosive premix is prepared at site where control/ supervision is more difficult.
- Complicated metering devices are required which restrict mechanical availability.

Repumpable EmulsionlSlurry and the Gelmaster System

In this system non-explosive premix prepared under controlled supervision at the plant or on the Gelmaster truck, is carried in a delivery truck to the mine site along with other ingredients. The explosive is mixed in batches and pumped down the borehole where mix attains its explosive properties afer a short period of time.

The Gelmaster system has several advantages over the earlier two systems, viz :

- No fixed explosive mixing plant is required.
- Non-explosive premix is transported at all stages ensuring maximum safety.
- Premix is prepard under better control supervision in a support plant and not at the mine bench, thereby allowing for necessary corrections to be taken if required before hand without jeopardising the blast.
- No moving parts inside the mixed bowl thereby rendering utmost safety in mixing operation.

- All drives are hydraulic/pneumatic eliminating possible safety hazards with mechanical drives.
- Variations in energy/density are possible to suit specific blasting conditions.
- No complicated metering devices, thereby improving mechanical reliability.
- Both slurries and emulsions can be delivered.

Silo Storage Facility

This is a unique storage facility of IDL's repumpable slurry or emulsion explosives. Non-explosive matrix manufactured at a facility is transported by mobile road tankers and transferred to 'Storage Silos' installed at the mining site. The pump trucks located at these sites draw their requirement from this 'Storage Silos'. Other requisite non-explosive materials are also stored in them.

Question 70 : Write short notes on ANFO/HANFO delivery system.

Answer : ANFO delivery requires an auger type delivery system. These units consist of a bin of AN, a fuel oil tank and an emulsion bin. The fuel oil and the AN are mixed in an auger to make ANFO. The emulsion is added in varying percentages in the boom auger of the truck to make the desired product. To get the right density, the emulsion and ANFO must be well mixed.

Question 71 : Discuss elaborately about the Gelmaster system.

Answer : **Gelmaster System Operational Features**

Equipment.

The heart of the Gelmaster system is the Gelmaster truck which contains the following equipment :

- Gelmaster mixer bowl which discharges into a blender where aluminium, if required, is added.
- Slurry/emulsion pump for pumping the product at high pumping rates (250 kg/minute).
- Flexible loading hose mounted on a retractable boom for mechanical hose handling.
- Equipment for dosing of liquid reagents to enable density and stability control.

Flow sheet of Gelmaster Truck is shown in Fig. 24.

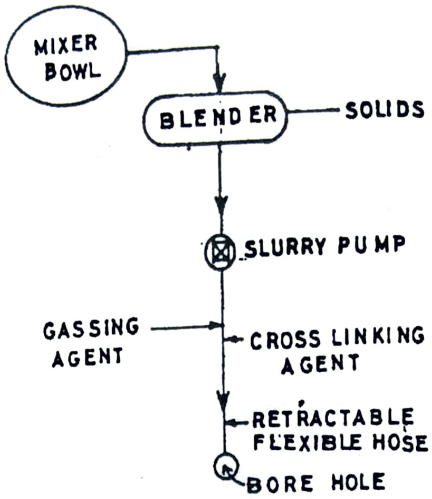


Fig. 24

Support Plant

The Gelmaster operates from a support plant which does not handle any explosive ingredients and hence requires aminimum of fixed on-site support facilities. It basically consists of storage for non-explosive raw materials and facilities for their processing into non-explosives intermediates. Thus no safety distance unlike fixed explosive plants are required and land required is 100 x100 m for 10,000 tpa plant. A typical layout of this plant is shown in Fig. 25.

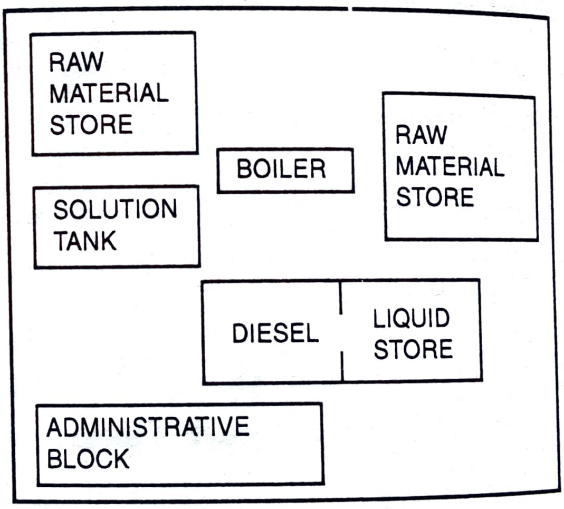


Fig. 25 Support Plant Layout

Operational Controls

The Gelmaster system incorporates batch type mixing, i.e. dry ingredients and liquid solutions for each batch are introduced into the mixing bowl from the support plant, mixed to exact specification, samples drawn, tested for quality and only when the batch meets the quality specification, truck leaves for the mine's bench to load boreholes.

At the site, all controls relating to on-line gassing, density control and other solids additions for energy variations are monitored from the cab and quantities delivered displayed on panels. Boom mounted

retractable loading hose is also operated from the cab by a single operator.

Question 72 : What may be the probable causes of accidents due to the explosives.

Answer : The probable causes of accidents which may occur due to the explosives are as following :

- Due to unsafe practices during storing, handling, and transporting.
- If the safety fuse is very short in length, and more time is taken to ignite a safety fuse.
- Careless handling of misfires.
- Drilling into the butt where unexploded explosive chargc may be present.
- Firing electrically the explosive charge in a careless manner without giving sufficient warning and also firing electrically before the preperation is complete in all respects.
- Returning to the blasting site before the stipulated time.
- Inadequate warming and inadequate posting of watchman before blasting.
- If tamping procedure is not correct.
- Smoking and lighting of any igniting materials at the vicinity of the explosive storage place or during handling of explosives.
- Not following the instruction of initiation of explosives from a proper and safe distance limit.
- Due to stray currents, electric storm, static electricity, lightning, etc.
- Due to charging improper explosives in hot holes; etc.
- Not taking proper shelter during blasting.
- By ignition of firedamp and coal dust which is caused by the incomplete detonation of explosive, incandescent particles coming out of the shot holes, production of flame by the explosives, blown out sho etc.

Question 62 : Write short note regarding the destruction of various explosives and blasting accessories.

Answer : Various types of explosives and blasting accessories can be destroyed in the following manner :

Gun powder

It should be thrown into the hot water where saltpetre will be dissolved and render the explosive harmless. An altenative method is to lay it out in a train and fire this from one end by means of a safety fuse, but if more than one trains are laid, care should be taken to lay the second one at such a distanace from the site of first as to run no risk of its being fired by the heated soil.

Ammonium Nitrate

Because these types of explosives make an excellent manure they may be scattered on damp soi1. Since they are very much hygroscopic in nature, as an immediate effect they may be buried in a ditch which is damp enough to render them harmless.

Gelignite, Getatine and other Nitroglycerine Based Explosives :

These types of explosives should not be destroyed more than 20 to 25 kgs at a time. A clear space of ground, free from any dry grass and inflammable substances, approximately 100 m all round, should be selected and a line of shavings of dry straw or grass is to be laid down on which the cartridges should be placed in a continuous line not more than two abreast the cartridge wrappers and any other available paper below them and with an interval of 30 mm between the two cartridges. Paraffin or other similar oil should then be poured over the shavings of grass or straw to accelerate the combustion. The line of the shavings of straw or grass should be prolonged some distance (7 m approximately) beyond the explosive and lit with a short length of safety fuse. The Operator should take a proper shelter beyond the danger zone after leaving the burning site. The wrapper of the explosive cartridges should be opened and unrolled to avoid any explosion.

Dynamite

Not more than 2.5 kg of explosives should be burnt at a time and the wrappers must be opened and unrolled. The site on which the destruction is to take place should be so chosen and the fire is to be initiated by means of safety fuse of such a length that no risk is run by personnel or property in the event of the fire charging to explosion.

Safety Fuse

After taking suitable precautions the safety fuses should be destroyed by burning in length in the open air.

Detonators

After disposing off to the deep river or to the sea, detonators should

be thrown into deep water by two and three or they may be thoroughly soaked in mineral oil for 48 hours and then destroyed by burning one at a time after taking suitable precautions.

Question 74 : Mention salient points regarding the transportation of maximum quantity of explosives by various means.

Answer :

(i) A person should not carry more than 5 kg of explosive in a sealed container.

(ii) Railway wagon can carry upto 10 tonnes of explosives or half the carrying capacity of the wagon whichever is less. Not more than 5 such carriages containing explosives shall at any one time be loaded or unloaded at or on any railway station or be attached to or transported by any one train.

(iii) The maximum amount of explosive is permitted to be carried by a boat is 22500 kg.

(iv) On road a maximum amount of 10000 kg or 75% of the carrying capacity of the van licensed by the Regional Transport Authority, whichever is less is permitted in one carriage. Provided that if the explosive to be transported is one of the second class, the quantity of explosive shall not exceed 15000 kg or 75% of the carrying capacity of the van, whichever is less.

(v) All mechanically propelled vehicles used for transportation of explosives shall be of a design approved by the Chief Inspector of Explosives and shall be locked irrespective of their carrying capacity.

(vi) Unless all iron or steel therein with which a package containing any explosive is likely to come into contact, is effectively covered with lead, leather, wood cloth or other suitable material, no carriage shall be used for transporting explosives.

(vii) Wen a carriage is used to transport explosives if explosives exceeds 1000 kg in weight, they shall be placed in the interior of the carriage which shall be enclosed on all sides with wood or metal so as to effectively prevent the explosives from communication of fire and the carriage shall be completely covered with the fire proofed cloth, tarpaulin or any other

suitable material so as to effectively protect the explosives from communication of fire.

(viii) If the explosives in two or more carriages or boats travelling together exceeds the maximum prescribed limit as mentioned above for any one carriage, a safety distance of minimum 50 m should be maintained between them, however this does not apply to railway wagon.

(ix) If any repairs or alterations are required in any part of a carriage or vessel in which explosives are being or have been transported, all due precautions shall be taken to remove all of such explosives or any remnants thereof and the space in which such explosives have been carried shall be thoroughly washed out to ensure that no remnants of explosives remain therein.

Question 75 : Discuss about the transport of explosives in bulk for the deep hole blasting in surface mines.

Answer : It is given as an amendment in the Reg. 164A of Coal Mines Regulation which is as follows :

1. Transport of explosives from the magazine to the priming station or the site of blasting shall not be done except in day light and in the original wooden or card board packing cases. Thc quantity of explosive transported at one time to the site of blasting shall not exceed the actual quantity required for use in one round of shots. The explosives shall be transported to the site of blasting not more that 30 minutes before the commencement of charging of the holes.
2. (a) No mechanically propelled vehicle shall be used for the transport of explosives unless it is of a type approved in writing by the Chief Inspector, provided that a Jeep or Land Rover may be used for the transport of detonators from the magazines to priming stations, subject to all the following conditions :
   (i) not more than 200 detonators are transported in a vehicle at a time;
   (ii) the detonators are packed suitably in a wooden box;
   (iii) the wooden box containing detonators is placed inside

an outer metal case of a construction approved by the Chief Inspector.

(iv) the outer metal case shall be suitably bolted to the floor of the vehicle or otherwise fixed in a wooden frame so that the container is not displaced while the vehicle is in motion, and

(v) no person shall ride on the rear portion of the vehicle.

(b) Every vehicle used for the transport of explosives shall be marked or placarded on both sides and ends, with the words 'EXPLOSIVES' in red letters not less than 15 cm high on a white background.

(c) Every mechanically propelled vehicle transporting explosives shall be provided with not less than two fire extinguishers (one of Carbon Tetra Chloride Type for petroleum fire and the other of Carbon Dioxide under pressure type for electrical fire) suitably placed for immediate use.

3. (a) The vehicle used for transport of explosives shall not be overloaded and in no case shall the explosive cases be piled higher than the sides of the body.

(b) Explosives and detonators shall not be transported in the same vehicle.

4. (a) No person other than the driver or his helper (not below 18 years of age) shall ride on a mechanically propelled vehicle used for the transport. of explosives.

(b) A vehicle loaded with explosives shall not be left unattended.

(c) The engine of a vehicle transporting explosives shall be stopped and the brakes set securely before it is loaded/or unloaded or left standing.

(d) A vehicle transporting explosives shall not be driven at a speed exceeding 25 km per hour.

(e) A vehicle loaded with explosives shall not be taken into garage or repair shop, shall not be parked in a congested place.

(f) A vehicle transporting explosives shall not be refuelled

except in emergencies even when its engine is stopped and other precautions taken to prevent accidents.

(g) No trailer shall be attached to a vehicle transporting explosives.

5. (a) Every vehicle used for transport of explosives shall be carefully inspected once in every 24 hours by a competent person to ensure that :

i) fire extinguishers are filled and are in place;

ii) the electric wiring is well insulated and firmly secured;

iii) the chasis, engine and body are clean and free from surplus oil and grease;

(iv) the fuel tank and feed lines are not leaking;

(v) lights, brakes and steering mechanism are in good working order.

(b) A report of every inspection made under sub-clause (a) shall be recorded in a bound-paged book kept for the purpose; and shall be signed and dated by the competent person making the inspection.

6, All operations connected with the transport of explosive shall be conducted under the personal supervision of an overman solely placed in charge of blasting operation at the mine.

7. The shotfirer shall personally search every person engaged in the transport and use of explosives and shall satisfy himself that no person so engaged has in his possession any cigar, cigarette, "biri", or other smoking material or any match or any other apparatus of any kind capable of producing a light, flame or spark.

Question 76 : Which are the explosives that can be stored together in the same magazine?

Answer : Under the explosives Rules the following types of the explosives can be stored together in the same magazine :

Gun powder	class - 1
Nitrate mixtures	class - 2
Nitro-compounds	class - 3
Chlorate mixtures	class - 4

Safety fuse class - 6, Div. - 1

Plastic igniter cord,

'Cordtex' detonating fuse class - 6, Div. - 2

Detonators should not be stored together with the above. However, detonators not exceeding 44,000 in number can be stored in a properly constructed annexe attached to the magazine but for a higher number a separate magazine is to be constructed.

Question 77 : What is a Magazine? Discuss.

Answer : A 'Magazine' is a building where explosives and detonators are stored.

The following points are very important regarding the magazine :

- It should be located in the remote isolated place, preferably in a jungle or hilly area, so that if any accident occurs the damage of the public property is least.
- The magazine should be connected with well maintained approach road throughout the year.
- The magazine should be situated away from the overhead power transmission lines at a minimum distance of 90 m.
- Magazine should be situated in a slopping ground where there is good drainage.
- The clearance distances around a magazine for storage of gunpowder and high explosives is given in the Table - 23.

Table 23

CLEARANCE DISTANCES AROUND A MAGAZINE FOR STORAGE OF GUN-POWDER AND HIGH EXPLOSIVES

Quantity of explosives	Distance between Magazine and room of workshop used in cennection with magazine; any other explosive magazine or store for explosives with the consent in writing of the occupier, Magazine office.	Distance between Magazine and Magazine keeper's or chowkidar's dwelling house; railway canal or other navigable water dock, pier or jetty; market place, recreation and sports ground or other place where the public are accustomed to assemble; public highways; private road which is a principal means of access to a	Distance between Magazine and dwelling house, retail shop; Govt. and public building, temple, mosque, church, gurudwara, or other places of worship : college, school, hospital, theatre, cinema or other buildings where the public are accustomed to assemble; factory; building or works used for the storage in

		temple, mosque, church, gurudwara, other places of worship, hospital, college, school or factory; river wall, sea wall, reservoir or bounded tank.	bulk of petrolium, spirit, gas or other inflamable or hazardous substances; builings of work used for the storage and manufacture of explosive articles which contain explosive; aerodrome; furnace, kiln or chimney quarrey or mine pit heads; power house; wireless station; warehouse or other buildings.
(1) kg	(2) Mounded m	(3) Unmounded m	(4) m	(5) m
181	13	20	24	48
272	16	23	32	63
363	17	26	38	77
454	18	27	45	89
907	23	34	70	140
1361	27	40	90	180
1814	29	44	107	215
2268	31	47	122	244
2722	33	50	134	269
3629	37	55	155	310
4536	394	59	172	344
6804	45		204	407
9072	50		227	454
13605	57		263	526
18114	62		290	581
22680	67		314	628
27216	72		334	668
36287	79		367	735
45359	85		396	792
68039	97		454	907
90718	107		499	998
113398	115		538	1076
136078	128		572	1134
158757	129		602	1204
181437	135		629	1259

- It should be well ventilated, strong, of bullet proof and fire resistant construction.
- There should be natural barrier like hills all around the magazine. However, if the natural barrier is not available then the magazine can be surrounded with artificial earth mounds on all sides. The mound should be kept free from shurbs, dry grass and other combustible material.
- To stock gunpowder or other explosives in quantities not exceeding 45 kg of gun powder, 5 kg of other explosives, 200 detonators or any quantity of safety fuse, a licence in Form 'J' is necessary. The licence is granted by the District Authority to whom the application is made.

The following conditions are attached to a licence in Form 'J'.

1. Quantity of explosives stored shall not exceed the licenced quantity.
2. The building where the explosives shall be kept should be constructed of non-inflammable materials separated from any dwelling house or other building, another licenced premises, highway, street, public thoroughfare or a public place by a distance of 50 m and made and enclosed so as to prevent unauthorised entry and to secure it from danger. Detonators must be kept entirely separate so as to prevent fire or explosion communicating from one to the other.
3. All substances or articles of an explosive or highly inflammable in nature shall be kept away from the magazine. Persons entering it shall not have iron or steel in their possession or attached to on their boots or shoes.
4. Tools, implements, etc. shall not be made of iron or steel.
5. The quantity of gunpowder or other explosive exceeding 450 gms (approx) shall be kept in a substantial bag, case or canister of proper construction.
6. Records showing issue and receipt of explosives shall be maintained as directed by the District Authority.
7. Any shortages, losses and thefts, etc. shall be reported to the nearest police station without. delay.
8. The licencee shall at the time of purchase have the following

particulars endorsed upon his licence and signed by the vendor from whom he purchased.

a) The name and address of the person who takes delivery of the articles purchased.

b) The nature and amount of the article purchased.

c) The date of purchase.

9. The licence may be filling cartridge, making charges or drying, fitting or otherwise adapt or prepare for use any explosive he is authorised to posses under this licence, provided that

a) There shall not be in the workshop in which such adaptation or preparation is carried on more than 45 kg of gun powder, or such quantity of other explosive as is prescribed by the licensing authority in this behalf.

b) The total quantity of explosive on the licensed premises including the workshop shall not exceed the quantity the licencee is authorised to possess.

c) Work not connected with such adaptation or preparation shall not be carried on in the said workshop while such adaptation or preparation is being carried on.

d) The workshop and the magazine should be separated and the former shall be situated in the immediate neighbourhood thereof and at such distance therefrom as may be prescribed in the licence.

e) A type of explosive of a certain description shall not be converted into an explosive of another description and an explosive shall not be unmade or resolved into its ingredients.

f) The authority who granted licence should be provided with notice by the licencee that he intends to carry on such adaptation or perparation as is allowed by this condition.

Grant of licence in Form L is done by the Chief Inspector of Mines to possess explosives (other than fulminates) in and to sell explosives as mentioned in 'Grant of Licence in Form 'J'. In this case for upto 22680 kg of possession and, possession and sale, fees of Rs.100 and Rs.125 respectively is to be paid and Rs.10 and Rs.12.50 for every additional 4536 kg, is to be paid subject to a maximum of Rs.250/- for possession and possession and sale respectively. Some of the important conditions

attached to the licence in Form L is given as following :

1) The total quantity of explosives stored at any time shall not exceed the licenced quantity.
2) Only the explosives of the kind mentioned in the licence and of receptacles or of tools or implements for work connected with the keeping of such explosives shall be kept in the magazine.
3) No opening of packages and the weighing and packing of explosives shall be carried out in the magazine.
4) It is permitted to keep two or more descriptions of explosives in a magazine but shall be kept separate by an intervening portion of such substance and character or by such intervening space as will effectually prevent explosion or fire in the one communicating with the other. Provided that the following explosives may be kept with each other without any intervening partition or space : Gunpowder (class-l), Nitrate Mixtures (class-2), Nitro compounds (class-3), Chlorate mixture (class-4), Safety fuse (class-6; Div.l), Plastic igniter cord, cordtex detonating fuse (class-6, Div.2). On no account detonators must be stored together with the above.
5) The interior parts of the magazine and the benches, shelves and fittings therein shall be so constructed or so lined or covered as to prevent the exposure of any iron or steel and the detaching of grit, iron, steel or similar substances in such a manner as to come into contact with the explosives. Such interior, benches, shelves and fittings, shall so far as is reasonably practicable, be kept free from grit and otherwise clean and in the case of any explosive liable to be dangerously affected by water, due precaution shall be taken to exclude water therefrom. However, this provision shall not be obligatory in a building in which only safety fuses are stored.
6) The construction of the magazine is such that it provides explosive to be kept therein and boxes containing explosives shall not be stacked more than 3 m in high.
7) The magazine should be provided with an efficient lightning conductor, which shall be tested atleast once in every year. All explosives shall be removed and the floor cleaned and washed before any repairs are done in the magazine or any part thereof.

9) All implements and tools used in any part of the magazine shall be made only of wood, copper or brass and some soft metal or material or shall be covered with some safe and suitable material. This condition does not apply to a magazine if only safety fuses are stored in it. In magazine only brass or gunmetal locks and keys should he used.

10) Only artificial lights apporoved by the Chief Inspector of Explosives should be used in a magazine. Ordinary dry cell torches are not allowed inside a magazine.

11) Smoking is not allowed in any part of the magazine premises.

12) The magazine shall be properly guarded as per the direction by the District Authority.

13) Any shortages of stock, losses and thefts of explosives shall be reported without delay to the nearest police station.

14) For preparation or adaptation condition is the same as condition (9) for the licence in Form J mentioned earlier.

Broadly magazines may be categorised into 3 types:

(1) Large magazine (2) Service magazine (3) Portable magazine.

Large magazine (Fig. 26)

i) The detonator store forms an integral part in this type of the

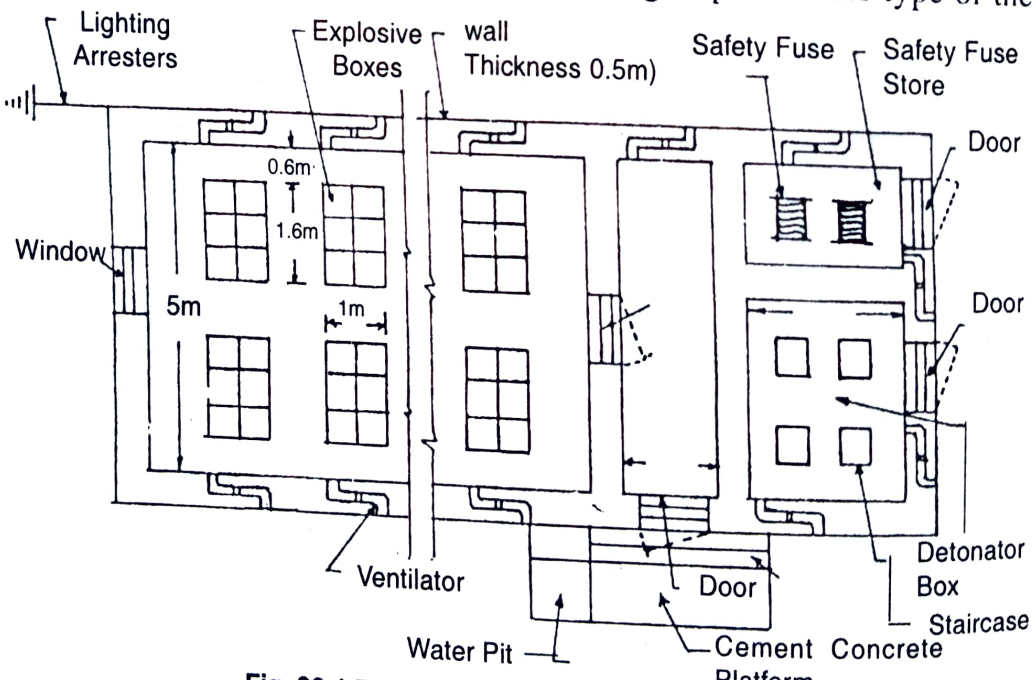


Fig. 26 A Typical Large Magazine for Explosives (Length depends upon the capacity) (not to scale)

magazine. The doors should be opened outwards and it shall be separated from the explosives store by a 50 cm thick brickwall and with an air space of minimum 1 m.

ii) The roof is to be made leakproof and may be made of R.C.C. Floor is made of 100 mm thick lime concrete with 25 cm cement plaster on top.

iii) The interior faces of all walls should be cement plastered and polished.

iv) Explosive magazine may be provided with a lobby.

v) All the doors and windows shall open outwards. The main door is made of 6 mm wrought iron plate faced inside with wood and has brass hinges and fittings. The inner doors and the door of the detonator annexe, and all windows are made of wood with brass hinges and fittings. Locking arrangements are to be provided to all the doors.

vi) The magazine is to be provided with Z type ventilators of size 228 mm x 114 mm in size fitted with a frame of iron bars set firmly in the wall on the outer face. The bars are 9.5 mm square in section set anglewise with the frame. The inner opening is to be protected with 8 mesh to the linear inch brass wire gauze. The ventilators are provided at a safe height.

vii) There should not be any electrical fittings in the magazine.

viii) The magazine should be provided with an effective lightning arrester and all iron and steel used in the doors, windows and ventilators are to be connected to this system, The ground plates of the system is to be buried 3.65 m away from the magazine building.

ix) Around each bench of explosive boxes a clear space of 600 m is to be provided. A space of 1.5 m^2 (better if it is 2 m^2) has to be provided for each tonne of the explosives stored.

x) For washing feet a water pit may he provided at the entrance of the magazine.

xi) No stack of explosives should be of more than 3 m in height.

xii) The magazine should always be kept securely locked and in the charge of a competent person. Persons shall not be allowed near a magazine during a thunder storm and magazine

should not be opened at that time. Explosives boxes should be opened only with the help of wooden wedges and mallets; and they should never be dragged or dropped on the floor nor the explosives boxes be opened inside the magazine. A proper record of all the issue and return of the explosives is to be kept in a bound paged book kept for the purpose. Explosives issued from the magazines shall be carried in closed containers or vehicles and all unused explosives shall be returned to the magazine. Damaged explosives should be taken out immediately for destroying in the approved manner.

Service magazine

This is generally prepared for the temporary storage of explosives not exceeding 227 kg in quantity and construction follows the most of the important norms as mentioned in the case of 'Large Magazine'.

Portable magazines

These are designed for storing upto 500 kg of explosives or 12000 nos. of electric detonators or 44000 nos. of ordinary detonator or 10000 m detonating fuse.

Both the large magazine and service magazine should be approved by the Chief Inspector of Explosives whereas the portable magazine should be approved by the Chief Controller of Explosives.

Calculation of the Capacity of a Magazine

To calculate the capacity of a magazine first of all find out the consumption of explosives/day and then find out the consumption of explosives in two months basis since two months' consumption of explosives are generally stored in a magazine. Now the calculation and decision will be as follows :

1) For Underground Coal or Metalliferous Mines

Let the daily production be X tonnes.

So explosives required per day = $\frac{X}{P}$ kg

Where P is the realistic production of coal/ore in tonne per kg of explosive.

Therefore explosives requirement for 2 months = $\frac{50X}{P}$ kg ..(1)

(Considering the number of days to be worked per month = 25)

With the value of equation (1) add up the quantity of explosives required for stone drifting/winzing/raising in stone for 50 days.

Say the value = *B* kg.... (2)

(Generally the stone work is restricted to 15 to 20% of the production in normal condition)

So equation (1) + (2) = $(\frac{50X}{P}) + B$ kg

$= \frac{1}{1000}(50\frac{X}{P} + B)$ tonne

So area of the magazine = $\frac{2}{1000}(50\frac{X}{P} + B)$ m^2

Here per tonne of explosive requirement of area considered = 2 m^2

Now design the magazine.

2) For Opencast Coal/Metalliferous Mines

Let the annual production of coal/ore = Y tonne

If the stripping ratio for that particular deposit is 1: 5; then overburden to be removed = 5 Y m^3 per year.

Let P_1 is the realistic production of coal/ore in tonne per kg of explosive.

And P_2 is the production of overburden in cubic metre per kg of explosive.

So consuption of explosive for 2 months :

For coal/ore = $(\frac{Y}{6P_1})$ kg ... (3)

For overburden = $\frac{5Y}{6P_2}$ kg ... (4)

So consumption of explosive for two months :

Equation (3) + (4) :

$= \frac{Y}{6P_1} + \frac{5Y}{6P_2}$ kg $= \frac{1}{1000}(\frac{Y}{6P_1} + \frac{5Y}{6P_2})$ tonne .... (5)

Considering the area of magazine for stacking explosives $2m^2$/ tonne of explosives.

The area of the magazine = – $\frac{2}{1000}(\frac{Y}{6P_1} + \frac{5Y}{6P_2})$ m^2

If the value of the equation (5) is very large then it will be a very

big magazine and requires a very large dimension of safe distance which is very difficult to acquire. Under these circumstances try to consider the main charge as LOX or SMS or ANFO or Emulsion Explosives for blasting ore/overburden along with the standard primer/booster. Because of this reason magazine for the conventional type of explosives will be very small and reasonable. Generally the requirement of booster charge is 5 to 25% of the total column charge. Find out the quantity of requirement of explosives and design the magazine.

Another way the problem can be solved is by making clusters of small magazines in the remote corners of the mine's lease-hold area. In this case the heavy quantity of explosives may be equally distributed among them. But in this case many complicated problems will be arising regarding the availability of the remote places, control, staffing, etc.

The other method to solve the problem is to construct a magazine initially considering the full capacity of the magazine but stock explosives only 25% of the final capacity of the mine since initial requirement of explosive is small. When the artificial mound by the overburden dump will be ready as the mine is in progress then obtain permission from the Chief Inspector of Explosives to store higher quantity of explosives.

Question 78 : Mention some salient points regarding the storage, transportation, mixing and charging of ANFO explosives.

Answer :

i) Ammonium Nitrate should not be stored with other explosives in the same magazine, but it shall be stored in a separate building made of non combustible or fire resistant material, floor of which is constructed with concrete. It should be equipped with closed type pipe drains into which molten Ammonium Nitrate can flow and be confined, in case of fire. The plinth level of the building shall be kept above the highest flood level and insides of the building shall be dry and clean. There should not be any electric wiring or heating arrangement inside the premises but interior should be kept cool and well ventilated.

ii) Ammonium Nitrate shall be stored in waterproof bags in a quantity not exceeding 50 tonne. No metallic substances in powder form, combustible material like fuel oils, acids, etc.

should be kept in the same storage premises. Ammonium Nitrate stacked piles shall not be more than 1 m in height and they shall be readily accessible in case of fire. To combat any accidental fire adequate supplies of water and fire fighting appliances should be readily available. Storage premises may be provided with an automatic water quenching system of sufficient capacity.

iii) Ammonium Nitrate bags shall not be damaged since the former is hygroscopic in nature and the loaded bags shall be used in the sequence of delivery to prevent accumulation of old stocks.

iv) Smoking, naked light or open flame shall not be allowed in the vicinity of Ammonium Nitrate, nor there should be any fire in the working areas. It should be stacked well clear of the diesel oil supply.

v) The unprocessed Ammonium Nitrate is transported to the blasting site with a vehicle (it must be in safe operating condition) driven by a competent licensed driver. No other explosives or combustible material including diesel oil should be transported in the same vehicle.

vi) For on site transportation of Ammonium Nitrate Fuel Oil mixture or for temporary storage of one day's consumption of explosive a well ventilated full trailer van may be used. Rubbish, dry grass or other combustible material for a distance of at least 9 m shall be kept free in all direction and smoking or open flames should not be permitted in its vicinity.

vii) Ammonium Nitrate shall be mixed with Fuel oil (in a quantity required for immediate use) in a wooden container (size: length 1.5 m, width 1.0 m, height 0.5 m approx) near the close proximity of the shot holes where approximately 100 kg of ANFO mixture can be prepared. The percentage of fuel oil in the mixture is around 5 to 6% which gives the maximum power. Excess of fuel oil percentage will lower down the sensitivity. The mixing should not take place in a building where Ammonium Nitrate is stored in bulk.

viii) Within 9 m of all around the mixing, charging, using or handling of ANFO mixture, there shall be presence of only

minimum number of required persons and any naked light or smoking is strictly prohibited. The site shall be also free from any rubhish, dry grass or other combustible material.

ix) The prepared mixture must be kept in water proof bags until the time of use and carried only in wooden container and the cap sensitivity of the ANFO mixture should be tested at a regular interval. The test is done in the following manner :

In a remote and isolated area a detonator is inserted into the smaller cartridge or other package of an ANFO mixture. After placing the cartridge or package inside a soft ground it is fired. If a crater is formed it will represent its cap sensitivity.

x) Unused ANFO mixture should be packed in inner (thick polythene bag) and outer (wooden or solid fibre board) packages and stored in a magazine duly approved by the Licensing Authority. The inner package must prevent the escapc of the explosive or oil.

xi) The ANFO mixture may be poured in free into the blast hole under the dry condition with the help of a suitable funnel of fibre board or wood, or may be poured into the blasthole in the form of packages of polythene bags for use in the wet hole. A wooden stemming rod may be used to aid the flow.

xii) Since ANFO mixture is less sensitive compared to the conventional high explosives, a primer cartridge of high explosive equal to 5 to 25% of the total charge shall be used. After blasting production of reddish brown fumes (which is an indication of highly toxic concentration of NO_2) may be observed and therefore a sufficient time should be allowed to disperse the fume before returning to the blast area.

Question 79 : Discuss about the Pulsed Infusion Shotfiring.

Answer : The pulsed infusion shotfiring technique essentially involves firing of an explosive charge in a borehole filled with water under pressure. When the charge is detonated it produces in the water a high pressure impulse which is transmitted into the numerous cleavage planes and slips, and thus breaks the coal.

A permitted explosive, 'Hydrobel' has been developed for this type of work. It fires satisfactorily after being subjected to a pressure of 68kgf/

cm^2 for 24 hours. The original submarine detonator was modified to produce the 'Hydrostar' detonator which is used exclusively for pulsed infusion firing. These detonators have thin tube bases, thick tube walls, high initiating power and 21 swg leading wires. Hydrostar short delay detonators are also available with delays ranging from 0 to 3.

Flanking Hole Method

Holes upto 2.15 m depth are drilled into the coalface at angles of 30^0 and 60^0, depending on the characteristics of the seam, to achieve an advance of 1.52 m. This method can be applied in both undercut and solid headings, longwall faces and depillaring work. The shotfiring procedure adopted is outlined below :

The infusion tube is first inserted in the borehole and the hole is closed by expanding the seal provided on the tube. The pump is started up for an initial infusion of the area surrounding the shothole. The pump is stopped and the tube is removed after water starts to seep out through the face. The explosive charge is then placed at the back of the shothole and the infusion tube is reinserted with the seal about 30 to 45 cm from the collar of the hole. Water circulation is resumed and the shot is fired with the water pressure still maintained in the hole.

For multi-shot or short delay blasting, it is necessary to have a manifold for supplying several infusion tubes simultaneously. A simple and robust manifold can be made at the colliery workshop.

Longhole Method

Specially designed drill rigs are necessary for boring horizontal hole of 30.5 m or more in length. Once a dependable technique of drilling has been established, this method may become very useful for solid blasting in depillaring and longwall faces.

The longhole method involves drilling one or two holes parallel to the face depending on the thickness of coal to be blasted. The burdens on holes vary according to seam characteristics. However, a 0.91 m slice can be dislodged per blast under average conditions.

The requisite number of cartridges are first taped into a length of detonating fuse with adequate spaces between the cartridges so that there is an even distribution of charge throughout the hole. For this type of work a special type of detonating fuse called 'Low Creep Cordtex' is used, which is capable of resisting water penetration through the ends at

high pressure. After the initial infusion has been carried out the, charge is placed inside the holes. One end of the hole is suitably plugged, the infusion tube is inserted at the other end and water pressure applied. A Hydrostar detonator is affixed to the Cordtex fuse inside the hole and only the leadwires remain outside the hole. The shot is fired in the same way as in the case of flanking holes. Fig. 27 shows the layout of a typical blast.

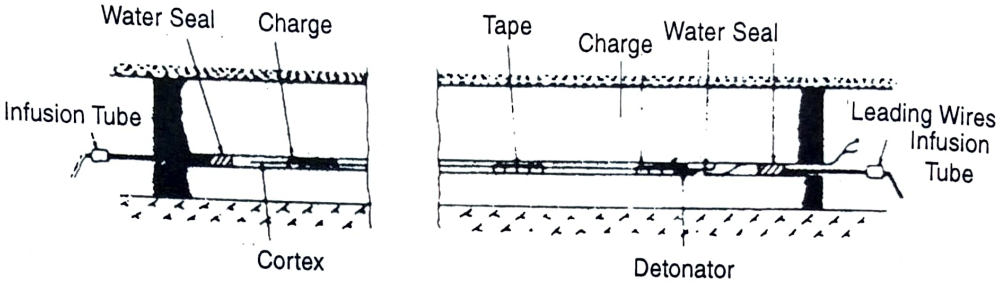


Fig. 27 Charging Longhole Employing Pulsed Infusion Shotfiring Technique

Question 80 : Discuss briefly about the Non-explosive rock breaking compound.

Answer : **Non-Explosive Rock Breaking Compound :**

It is a chemical compound comprising of lime (CaO), binding agent and retarder. The lime (expanding agent) is mixed with proper proportion of water and poured into the bore holes made into the ore/rock to be broken. Due to hydration of the expanding agent (CaO), its volume will increase twice the initial volume and will create a pressure inside the borehole. Due to this pressure first cracks will form around the holes, propagate and widen, which will ultimately fragment the rock/mineral. The whole process does not produce any noise, ground vibration, fly rocks or any gas. Due to hydration of lime, micro crystals of $Ca(OH)_2$ is formed and deposited over the surface of the grain of CaO and crystals. As massive aggregate of $Ca(OH)_2$ forms over the CaO extensive pressure (value may be upto 9kN) is developed which forces rock to break.

The duration of the whole process may take 16-24 hours. Expanding agent when mix with 35 to 40% water ratio, can break effectively (after 16 to 24 hours of use) a rock mass of compressive strength of around 700 to 800 kg/cm^2 and tensile strength of 35 to 60 kg/cm^2 while the spacing of the holes may be 10-15 times the diameter of hole.

The system can nicely be used for control blasting presplitting, blasting of rock/ore in the vicinity of the township or residential area,

secondary breaking of boulders, demolition of concrete, breaking of rock for road construction, breaking of quartzite and granite boulders.

Question 80 (A) : What is plasma blasting?

Answer : **Plasma Blasting**

An electrolyte solution is poured into the blasthole which thereafter transformed to high temperature-high pressure plasmastate by the supply of thousands MW electrical energy through specially designed coaxial electrodes placed inside the blast-hole. After the change of electrolyte to plasma, the latter expands at a faster rate compare to the normal seismic wave propagation velocity and forms a shock wave in the surrounding rock mass. The plasma blasting offers no fly rock, no ground vibration, no noise, produces very less amount of dust, results smooth face and the rock breakage cost by it is the minimum.

The plasma transformed from the electrolyte sloution in a borehole reaches an energy of 280 KJ by an electrical input of 8.5 KV and 200 Amp which rapidly releases energy into the borehole developing a shock wave which in turn produces a stress field that fractures the rock. The energy output may be to a level of mega jule and this energy is capable to fracture 1.5m high bench or $3m^3$ granite boulders. The plasma strength varies with chemical additives.

Question 80 (B) : What is B-Gel?

Answer : It is a non-explosive product developed in Canada. B-Gel when used in combination of low explosive energy material like detonating cord, it releases energy which split easily the parting between closely spaced contour holes drilled in an excavation. This B-Gel only assist failure of rock in a controlled manner which will definitely not produce the spider cracks. B-Gel when mixed with water yields the following characteristics—

- density : 0.95 gm/c.c.
- pH value : 9 to 10
- loading density : 1.94 kg/m in 50.8 mm diameter blast-hole
- non explosive
- non-toxic

This type of non-explosive product will assist for superior wall control in long hole stoping in metalliferous mines causing very less

dilution of ore with the country rock, and drifting walls and roof can be made smoother which will definitely reduce support requirement which in turn will reduce resistance of air flow. This product may be very much useful in dimension stone mining.

Question 81 : What is non-explosive rock breaking compound'? How it acts in the mechanism of rock breaking? Explain. How much time the system takes to break the rock? What maximum compressive and tensile strength of rock can it break'? Where this system can be used?

Answer : Consult the answer to question No. 80.

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Chapter 2

Blasting Practices in Surface Mines

Question 82 : What are the basic objectives of the blasting in surface mines? Where and why is controlled blasting is practised?

Answer : The basic objective of drilling and blasting program is to achieve optimum fragmentation. Blasting in overburden is mainly done either to fragment and shatter the rock or to displace the rock in the mine, area by casting of overburden. Blasting in coal is done only for fragmentation. Conventional or controlled blasting methods are adopted depending upon the proximity of the built-up area from the blasting site. Recently controlled blasting techniques have been successfully used to excavate coal on fire and also for the technique of presplitting to control overbreak and back break.

Blasting in surface coal mines is broadly divided into two parts :

1. Blasting in overburden
2. Blasting in coal

In both the cases either conventional blasting or controlled blasting is practised. Control blasting is done to control ground vibration, fly rock movement, air blast or noise, control of back break or overbreak and also to blast in the fire zone. For casting overburden also controlled blasting is done.

Question 83 : What are the factors to be taken into account during blast design? Explain.

Answer : The following are the some of the important factors which generally govern for optimal blast design :

- Physico-mechanical properties of rock

Here type of the rock, dynamic tensile strength, tensile strength, compressive strength, young's modulus, poisson's ratio, density and hardness of the rock mass, presence of discontinuities, bedding plane and joints, etc. are very important.

- Geology
- Pit geometry

Under this heading thickness of coal seam or ore body and bench height, overburden bench height, bench slope angle, strip width, height to

width ratio, length to width ratio are generally considered.

- Explosive characteristics –

  Factors generally considered under this heading are type of explosive, type of booster, bulk strenth, energy release per unit mass of explosive, detonation pressure, explosion pressure, ratio of decoupling, strength of explosive used, time taken for explosive wave to travel to the free face and back, volume of gaseous product per unit mass of the explosive, velocity of detonation, velocity of explosion propagation, explosion wave length, weight strength, number of spalls that an explosive wave may produce, length, diameter and weight of the cartridge, loading density, bottom charge and column charge density, etc. are very important.

- Characteristics of blasting accessories – type, thermal properties, etc.

- Burden distance
- Spacing of the hole
- Ratio of spacing to burden
- Depth of hole
- Diameter of blast holes
- Consideration of toe and depth of sub-grade drilling
- Blasting technique –

  Here objective of blasting, drilling pattern, number of availability of free faces, manner of charging, charge per hole and per delay, sequencing of initiation i.e. delay between two holes in a row and delay between two rows, decking, length of explosive cloumn, height of the bottom charge, volume of the explosive in the blast hole, etc., are to be considered.

- Powder factor

  The size of the fragmented rock should match the bucket size of the excavator and also the grizzly size of the primary crusher.

- Length of stemming column, the size and quality of stemming are very important factors.
- Proximity of structures

- Angle drilling
- Amount and direction of throw requirement and problems of fly rock.
- Requirement of muck profile
- Vibration level
- Presence of water, etc.

Question 84 : Enumerate the factors which govern the optimum blast designing in the surface mines.

Answer :

Factors Governs the Optimum Blast Design in the Surface Mines are as follows :

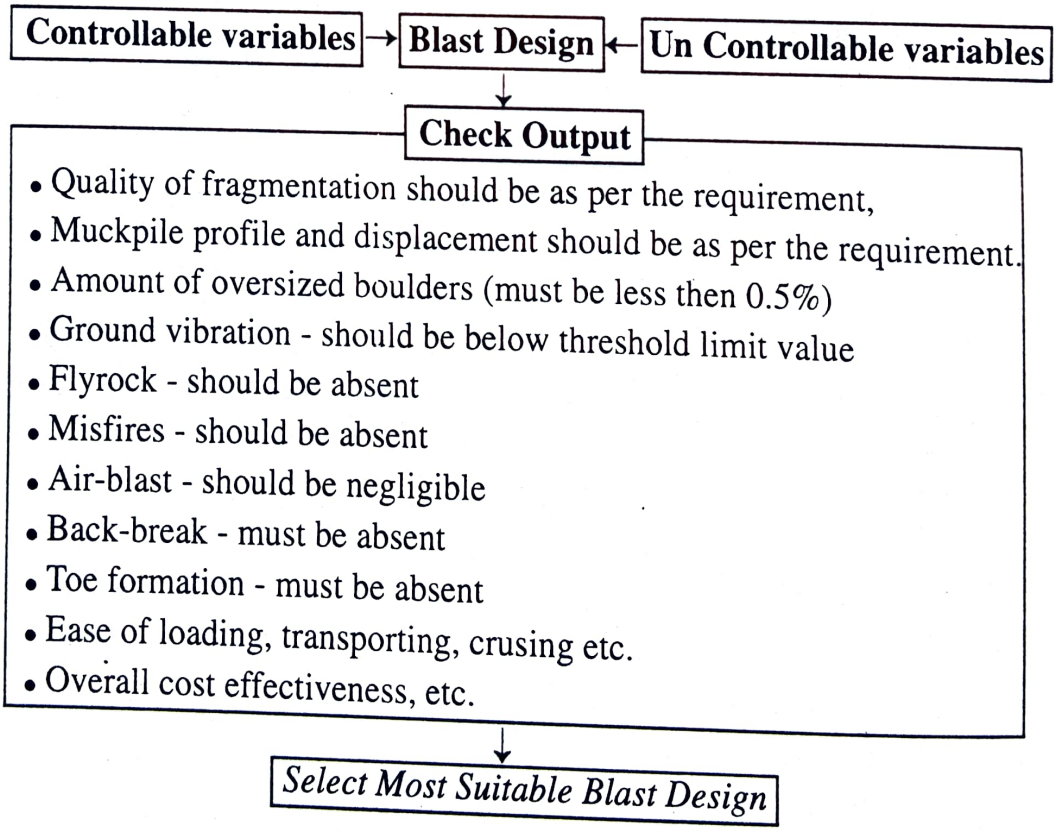
Controllable Variables

- Hole depth
- Hole diameter
- Subgrade drilling
- Hole inclination
- Stemming height
- Stemming material
- Bench height
- Bench width
- Bench length
- Blast pattern
- Burden to spacing ratio
- Burden to hole diameter ratio
- Hole diameter to stemming height ratio
- Blast size and configuration
- Directional blasting
- Availability of number of free face
- Initiation system and sequence
- Presence of water
- Buffers
- Explosive type and energy,
- Controlled blasting
- Deck charging

- Loading method/machinery
- Cost of drilling and blasting
- Requirement of frequency of blasting
- Availability and use of modern blasting accessories including exploder/firing system
- Trained manpower
- Management, supervision and control
- Location of blasting zone (near colony, hospital, township, etc)
- Impact in environment

Uncontrollable Variables

- Geology and site specific conditions
- Geomechanical parameters of rock/mineral body e.g., compressive strength, tensile strength, shear strength modulus of elasticity, poissions ratio, etc.
- Presenc of structural discontinuities, cracks, fissures etc.
- Presence of weaker strata in the rockmass
- Interaction of weather
- Water



Question 85 : What are the factors which are to be considered for the following :

a) Explosive characteristics

b) Physico-mechanical properties of rock

c) Blasting technique

Answer : Consult the answer to question no. 83.

Question 86 : What considerations are to be looked into for cost effective blast- hole design? Explain regarding measures that are to be taken for (a) prior to drilling (b) while drilling and (c) while charging the holes?

Answer : For cost effective blast hole design several considerations are to be looked into for optimum blast hole utilisation and measures need to be taken in the following phases of the drilling and blasting operation :

- Prior to drilling
- During drilling
- During charging of the holes

Measures to be Taken Prior to Drilling

For reducing the explosive consumption, special attention is to be paid on site geology, geotechnical parameters of rock mass and the selection of the direction of movement of the blasted material. Joints, fractures, dip etc. are to be considered in depth for blast hole design. Blast movement in the dip direction will reduce the need of amount of secondary blast but it may cause back break. While movement towards rise will reduce the amount of back break but it will result inferior blasting, whereas movement along the strike may create uneven floor and zigzag final slope. Direction of movement of the blasting product which depends upon the availability of number of free faces, can be predicted and controlled prior to drilling operation. But if the availability of free faces in all the part of the block to be blasted are not equal then damage of the face (where the free face is significantly reduced) will be more. Free face is also required to avoid the formation of big boulders and toes in the constrained part.

Measures to be Taken While Drilling

In blasting operation the selection of proper drill sizes and drill machines are very important. With the increase in the diameter of the

drill hole drilling cost is decreased accordingly. But during selection of the greater diameter of blast holes the factors like cost of blasting during changes in loading, hauling and crushing cost are to be considered in depth.

Now a days larger diameter bits are available to suit a drill machine which can produce large diameter chambers at the bottom of smaller diameter blastholes by under reaming operation. The large diameter blastholes merits less development work and smoothen bulk handling of explosives. Before starting any drilling operation markings are to be given for blast-holes confirming the designed blast-hole spacing and burden which should be modified with the changing strata conditions, provided that the adequate informations regarding the geotechnical parameters are available. The length of the hole which dictates the blast volume is very important since a slight reduction of length will cause more explosive consumtion and reduction in production.

Measures while Charging the Holes

For improving safety and lowering down the explosive consumption, various measures and precharging observations are required. A thorough investigation is required regarding the effect of previous blasts on front row holes, water accumulation in the holes and variation in drilled depth of the holes. Formation of cavity in and round the hole and other necessary informations can be obtained by sensing the drilled hole by mirror reflection. Remedial measures should be taken wherever required before charging the holes.

Beside above careful considerations are to be taken during and after charging operation regarding the following aspects :

Matching of Diameter of Explosive

Skin to skin contact of explosives with the surface of hole is required because any gap between explosive and drill hole wall reduces the explosion energy imparted on the hole surfaces. This explains why slurry explosive is more effective than cartridge explosive.

Stemming

Improper confinement of explosives can lead to wastage of explosives energy, and can also result in air blast, fly rock and boulder formation.

The most effective stemming material is sand, where interlocking particles provide maximum resistance to ejection. The normal practice, however, is to use drill gummings for stemming, as they are readily available at the hole collars.

Stemming length of about 0.7 to 1.1 times the burden is usually optimum. However, stemming length would depend on the nature of the strata and the point of initiation of the charge. Optimum stemming increases the blasting efficiency, reduces the amount of explosives requirement per blast, reduces ground vibration, air blast, fly rock and formation of boulders and also improves fragmentation of rock. Angular type pebbles form the best stemming material since they wedge themselves against the wall of the boreholes as the gas pushes them with high pressure during expansion.

Selection of Fuse and Boosters

Purposeful selection of B-cord down line can reduce energy loss upto 50%. As the percentage of cost of fuse in blasting operation is negligible a proper selection should be ensured even if it costs more. Because, that helps in reduction of total blasting cost per unit volume of broken material.

Besides the proper selection of fuse and booster, their placement in the hole is an equally important aspect. The adverse effect of improper placing may cause partial explosion or under-utilised explosive column. This results in increase of burden for surrounding holes and thereby it needs more explosive consumption and a use bad blasting.

Right Choice of Delays and Proper Placing of Delay Materials

A proper delaying sequence can not only reorient the face but also successfully obviate other difficulties of the site like non availability of proper free face, etc. Generally the effect of long inter row delays is that the burden in front of each successive row of holes begins to move before the subsequent row of holes is fired. This shows that each row of holes gets an effective free face when it is fired producing a good surface heave and leaving a trough at the rear end of the blast. As the explosion energy is more effectively utilised there is a significant reduction in fly rock generation. Sequencing of the blast-hole, when applied, reduces blast vibrations.

Question 86 (A) : Deduce some emperical equations for blast design.

Answer :

Some Empirical Equations on Blast Design

1) By Fraenkel (1944)

$$B_{max} = \frac{K}{50}\left(\frac{d^8}{3h_cH}\right)^{\frac{1}{3.3}}$$

where

B_{max} = Maximum burden for good fragmentation, m
d = Borehole diameter, m
h_c = Charge height, m
H = Depth of the blasthole, m

2) By pearse (1955)

$$B = \frac{Kd\sqrt{Ps}}{\sigma t}$$

where

K = a constant. It's value varies from 0.7-1.0
Ps = Borehole pressure, MPa
σt = Tensile strength, Kg/Cm^2
d = Diameter of borehole, m

3) By Langefors and Kihlstrom (1968)

$$B_{max} = 0.958\, d\sqrt{\frac{\rho e\, Se}{(S/B)\, C_0 f}}$$

where

B_{max} = Maximum burden for good fragmentation, m
d = diameter of hole, m
ρe = Density of the explosive in the borehole, Kg/m^3
Se = Weight strength of the explosive
f = Degreee of confinement of the blasthole.
S/B = Spacing to burden ratio
Co = Corrected blastability factor (Kg/m^3)
$= C + 0.05$ for $B_{max} = 1.4 - 1.5m$
$= C + 0.07/B$ for $B_{max} < 1.4m$
When C = rock constant

4) By Konya and Walter (1990)

$$B = 3.15 de \sqrt[3]{\frac{\rho e}{\rho r}}$$

$$B = \left[2\left(\frac{\rho e}{Pr}\right) + 1.5\right] de$$

where

B = Burden, (ft)

ρe = Specific gravity of explosive, (lb/in^3)

ρr = Specific gravity of rock, (lb/in^3)

de = Diameter of explosie, (in)

Correction factor

Bc = Kd. Ks. Kr. B

Where

Bc = Corrected burden (ft)

Kd = Correction factor for rock deposition. It's value is as follows—

- for bedding steeply dipping into cut Kd = 1.18
- for bedding steeply dipping into face Kd = 0.95
- for other cases Kd = 1.0

Ks = Correction factor for geologic structure. It's value is as follows—

- for heavily cracked, frequent weak joints, weakly cemented layers Ks = 1.30
- for thin well cemented layers with tight joints Ks = 1.1
- for massive intact rock Ks = 0.95

Kr = correction factors for number of row. It's value is a follows—

- for one or two rows of blastholes Kr = 1.0
- for third or subsequent rows Kr = 0.95

Konya and walter suggested the following emperical relationships—

- for instantaneous initations system $S = \frac{L + 2B}{3}$, $S = 2B$

- for delay initation system $S = \frac{L+2B}{8}$, $S = 1.4B$

where L = depth of blast-hole, m

B = burden, m

S = Spacing, m

Konya and Walter also suggested the following emperical relationship—

$$B = 0.67d(S_{ANFO} / \rho r)^{0.33}$$

where

S_{ANFO} = relative strength of explosive (bese ANFO)

ρr = density of rock, gm/c.c.

d = diameter of blast-hole, m

Question 87 : Write short notes on the following :

a) Matching of diameter of explosive

b) Stemming

c) Selection of fuse and boosters

d) Right choice of delays and proper placing of delay materials.

Answer : Consult the answer to Question 86.

Question 88 : Why directional blasting is practised? What is air drag? How does it relate with the casting factor?' How does the size of fly rock relate with air drag and casting factor? Why does slab charge suit the directional blasting? explain.

Answer : Overburden casting by the directional blasting offers a number of advantages which are as follows :

1) Saving in time and capital and operating cost in rehandling the blasted material since it is not to be transported.
2) The every cycle of operation commences early which results in increase of overall production and productivity.
3) Production will increase without the deployment of additional mining equipments.
4) In case stripping ratio goes up in a pitching seam, the same consistant production may be achieved with the deployment of same mining equipments.

5) Since the directional blasting arranges a predetermined spoil pile, it does not much deteriorate the environment.

Directional blasting involves a very complex interacting factors which include drilling and explosive aspects, casting factor, air drag, surface topography of the blasting site and spoil-dump, the distance between the blasting site and spoil-dump, economic consideration, etc.

The factors already mentioned are dependent on a very important factor like "Air Drag", which is the resistance to the movement of rock fragments while in trajectory and is again, compounded by interacting components such as linear size of the flyrock, its initial velocity and the angle that its trajectory takes with the horizontal. The affects of air drag in directional blasting which is mentioned below is very important

1) To locate the site for the spoil-pile where the product of directional blasting will move.
2) If the site for spoil-pile is decided then it determines the casting factor.

When casting factor is poor then a part of the product of directional blasting or the blasted overburden does not get cast (reach) at the predetermined spoil-pile and hence there would be the need of rehandling by shovel and dumpers and the inefficient practices will affect the cost factor adversely.

The size of flyrock is another important factor for air drag and casting. If the density, initial velocity, angle of trajectory to the horizontal remain same but the sizes of the blasting product is different then flyrock will fly in different distances while casting. For example at a flyrock density 2.7 tonne/m, with an initial velocity of 76 m/s and angle of trajectory to the horizontal 45^0, the maximum flying distance for 50 cm size may be 300 m. The distance of travel of flyrock may even go to more than 600 m. Sizes of flyrock varies over a wide range. Larger the size longer the distance it travels and hence the fragments on the spoil-pile scatter in the order of their increasing size away from the site of blasting.

The initial velocity at which the blasted fragments fly in the trajectory is another very important factor. Since air drag works inversely with the size of flyrock, slab charge suits the directional blasting and provides necessary energy input for the compact blasted mass to set off in the trajectory as desired. The slab charge improves the directivity and

casting factor (as high as 90%). When the angle of inclination with the horizontal is more than 20^0 the entire rock mass above the slab charge is cast on the spoil-pile. But in directional blasting the thickness of the charge should be considerably small compared to its length and breadth. So the slab charge is replaced by deep hole charges placed in the same plane at close intervals. The distance between the charge is limited to a value such that the compression wave from each charge is merged into each other which causes the resultant wave to travel parallel to the plane of the charges. But if the distance between the charges exceeds the limit, the blasting will result in a set of alternate humps and depressions.

To make directional blasting simpler ballistic tables can be prepared with the help of complex mathematical model worked out on a computer with a wide range of input parameters. This will help the designer to calculate quickly the trajectory of flyrock and will have an idea about the configuration of the spoil-pile. Nomographs which are drawn with strain energy based on type of rock, powder factor, the distance between rock mass where blasting is to be done, the site of spoil-pile, diameter of blast holes, loading factor of the charged holes, total charge weight per hole, length of holes loaded, explosive density, etc will help designing casting of overburden by directional blasting.

Question 89 : What is blast casting? What advantages may be obtained from blant casting? What are the parameters which govern cast blasting? Give an example.

Answer :

Blast Casting

Blast casting of overburden is the methodology of displacement of portion of overburden rock horizontally by blasting to a desired direction and distance away from the working face. Hence it is basically a directional blasting. The rest portion of the overburden is handled by the dragline/shovels.

Generally the fragmented overburden is removed over the demineralized area.

Advantages

- Saving in operational, maintenance and capital cost of excavating equipment for overburden removal. Only a little amount of money is necessary for explosives for overburden

casting which is nominal in comparision of expenditure for Heavy Earth Moving Machinery. Hence overall economy is achived in removal of overburden by cast blasting.

- Very less time is necessary for overburden removal. Hence production and productivity is increased with this technique.
- After side casting is over, for removal of the rest of the overburden low capacity excavators are necessary. Requirement of manpower, spares, and inventory management are also nominal in cast blasting. Hence it will assist overall economy of the mine.

The parameters which govern the cast blasting (or blast casting) are as follows :

- Physico-mechanical properties of overburden rocks, presence of discontinuities, geological disturbances, water, etc. in the strata layers. Blast casting is suitable in medium to hard rock condition when heavy blasting can be done for fragmentation. But blast casting be done in loose and unconsolidated rock.
- Optimum blast-design : Under this heading parameters like spacing, burden, diameter and inclination of blastholes, initiation sequence and delay intervals, etc. are very important. The spacing and burden shall be selected in such a manner that the maximum ejection velocity and fragmentation are achieved. Similarly the stemming height should be such that proper explosive energy is utilized for fragmenting the rock and for achieving the maximum ejection velocity. The blast holes must be inclined for achieving better throw. The higher the height of bench more percentage of materials will be thrown into spoil dump. For better result in blast casting bench height and bench width ratio shall be kept between 0.8 to 1.0.
- Types characteristics of explosives : Explosives having heaving characteristics are the best proposition for cast blasting. Hence ANFO or Heavy-ANFO explosives are the best explosives for cast-blasting. Higher specific charge by a factor of 1.3 to 1.5 times the conventional blast is the best for blast-casting. The basic criteria of cast blasting is the throw of rock mass row to row without any hindrances (like air

collisions, or flight collisions of the fragmented rock mass). Hence initiation of sequences and selection of delay intervals are the one of the most important criteria in cast-blasting. Subject to the allowable ground vibration limit a delay intervals of 10 ms/m of burden for the first 4 rows and a delay intervals of 20 ms/m of burden for the rest of the rows are the best proposition.

Example of Cast-Blasting

- Name of the Colliery : Sasti Opencast Mine, WCL.
- Strata : Medium hard sandstone (compressive strength varies from 120 to 220 Kg/cm^2
- Average Burden (m) : 9
- Average Spacing (m) : 9
- Average depth of boreholes (m) : 32.5
- Average height of bench (m) : 32.5
- Average height of water in the blasthole (m) : 12
- Number of holes blasted in a patch : 49
- Number of rows : 4
- Diameter of blasthole (mm) : 260
- Average height of stemming (m) : 7
- Sub-grade drilling (m) : Nil
- Explosive used : i) Site Mixed Slurry-MEXBLAST-B
  VOD : 4000 ± 200,
  Dendity 1.22 gm/c.c
  Water resistant : excellent
  It is the main column charge.
  ii) Booster explosive-MEXPRIME.
  It is 4% aluminized cartridge slurry booster charge. VOD : 4200 ± 200,
  Density : 1.2 gm/c.c
  Water resistant : excellent

Quantity used : 10% of the column charge.

iii) Cast booster-Mac Pentolite 50:50 of TNT/PETN

About 0.2% of the column charge.

- Average charge/hole (Kg) : 1543.35
- Delay intervals : i) Between 1st and 2nd row : 100 ms
  ii) Between 2nd and 3rd row : 125 ms
  iii) Between 3rd and 4th row : 150 ms
- Total blasted rock (m^3) : 128992.5
- Powder factor (m^3/kg) : 1.70
- Result : A portion of the overburden was well thrown in the decoaled area. Maximum depression observed in the high wall side was 17 m from top and a minimum depression of 9 m was observed in other place. The average depression over the area of the blast was found as 1 m from the top. 49% of the total blasted over-burden was casted by the blast. It was highly cost effective technique for the mine.

Question 90 : Enumerate rules of thumb regarding the dedign of blast in case of Blast-casting.

Answer :

Thumb Rules for Blast-Casting

For blast casting the following rules of thumb may be followed :

Bench height : 12 to 45 m

For satisfastory throw

Bench Height/Burden ratio : > 2.5

For Optimum breakage

Bench Height/Burden ratio : 2.5

The optimum Bench Height/pit width

		ratio lies in between 0.5 to 1. However Bench Height/Pit width ratio varies from 0.3 to 2.3
Explosive used	:	ANFO, and Heavy ANFO since they have higher heave energy compare to strain energy.
Powder factor achievable	:	0.6 to 1.1 Kg/m^3
Diameter of blast-hole	:	160 to 311 mm Common : 270 mm
Blasthole height	:	18-20 times the blasthole diameter or 0.6 to 1.4 times the burden.
Burden/Spacing ratio	:	0.5 to 1.0
Inter hole delay	:	9 ms or 17 ms
Inter row delay	:	for first 3 to 5 rows - 100 ms for other rows : 25 ms with an increment of 25 ms for every row.

Question 91 : What advantages are offered by inclined hole drilling? With the help of line diagram show a generalised pattern of hole with sequence of initiation in a 10 m bench of coal measure rock.

Answer :

Advantages of Inclined Hole Drilling

The advantages of inclined hole drillings are as following :

- more fragmentation and displacement is possible,
- eliminaie effectively front row toe burden and digging problem,
- probability of cut offs is lower,
- benches of higher thickness can be worked effectively and efficiently,
- produce less amount of overbreak,
- new face created by inclined hole drilling is smooth and sound,
- produce less vibration,
- reduce secondary blasting cost and also reduce consumption of explosives,

- cost of drilling, blasting, loading and crushing is reduced.
- angle of inclined holes generally varies from 15^0 to 30^0.

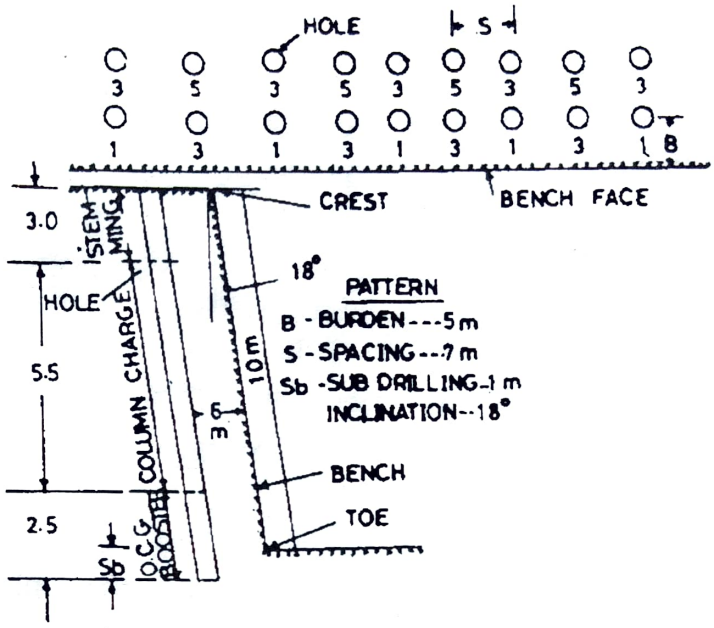


Fig. 28 Shown a generalised pattern of hole with sequence of initiation in a 10 m bench of coal measure rock.

Fig. 28 shows inclined hole drilling pattern with sequence of initiation. In the figure at the bottom of the holes booster charge used is OCG whereas ANFO is used as column charge. Initiation can be made by detonating fuse-trunk line laid in multiple V-cut pattern or other pattern and can be initiated by milli-second delay detonators. Both down and trunk lines are in pairs and both trunk lines of detonating fuse are tied together all along the length to take care of discontinuity in one by other to eliminate misfires.

However, the following demerits are associated with the inclined hole drilling.

- Drilling accuracy decreases as the angle of drill holes increases and as a result there may be the chances of much deviation in optimum burden and it leads to decrease in blasting efficiency drastically.
- Difficult in closer supervision. However, with the use of sophisticated drill machine this problem may be solved.
- There is an increased risk in borehole collapsing.
- Difficult of loading explosives and booster charge into the blasthole.

- Difficult to put detonating fuse into the blasthole.

Question 92 : Discuss briefly regarding the importance of drill pattern.

Answer : During drill pattern local geology, location of free faces, shape of the area, fragmentation planned, dimension of boreholes and explosives, type of explosives, depth of holes, local blasting experience, etc. play important roles. An improper drill pattern will lead blownouts, fly rock, air blast, heavy ground vibration, poor fragmentation, over break or under break, toe formation, wasting of explosive energy, strains on the loading machine, secondary blasting and overall uneconomy. Basic drill patterns are square, rectangular and staggered. Other drill paterns are chosen based on the manipulation of these patterns.

Question 93 : Why is single row blasting pattern preferred over the multiple row blasting pattern'? With the help of line diagram show the plan of sequence of initiation in single row blasting.

Answer : In opencast coal mines both the vertical or inclined holes parallel with the bench face is practised. Rows of the holes may be in single or multiple. Multiple row blasting pattern gives better fragmantation but specific explosive consumption is lower and hence more economic. Because of this reason in most of the coal benches and coal measure rocks single row blasting is preferred.

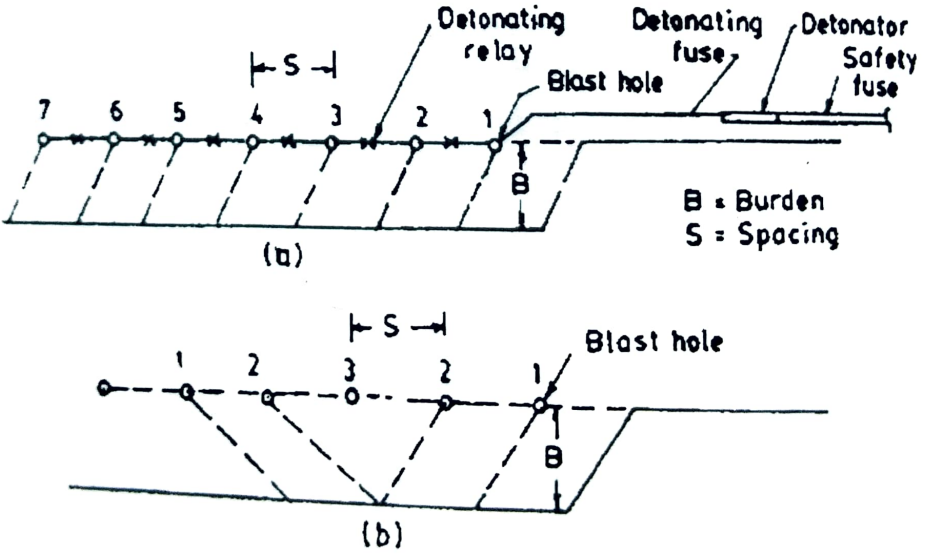


Fig. 29 (a) (b) Sequence in initiation in Single Row Blasting

Fig. 29(a, b) shows the plan of sequence of initiation in single row blasting with milli second delay detonators.

Question 94 : Design various multi row firing pattern.

Answer : **Multi Row Firing Pattern**

In multi row blasts, while in-line firing systems are also used, a large number of other patterns are available which provide better fragmentation and blasting efficiency by varying the effective burden B_E and spacing S_E. Some examples are shown in the fig. 32. Fig. 32(b) shows the 'V' shape delay initiation pattern using cord elays of 17/25 ms delays.

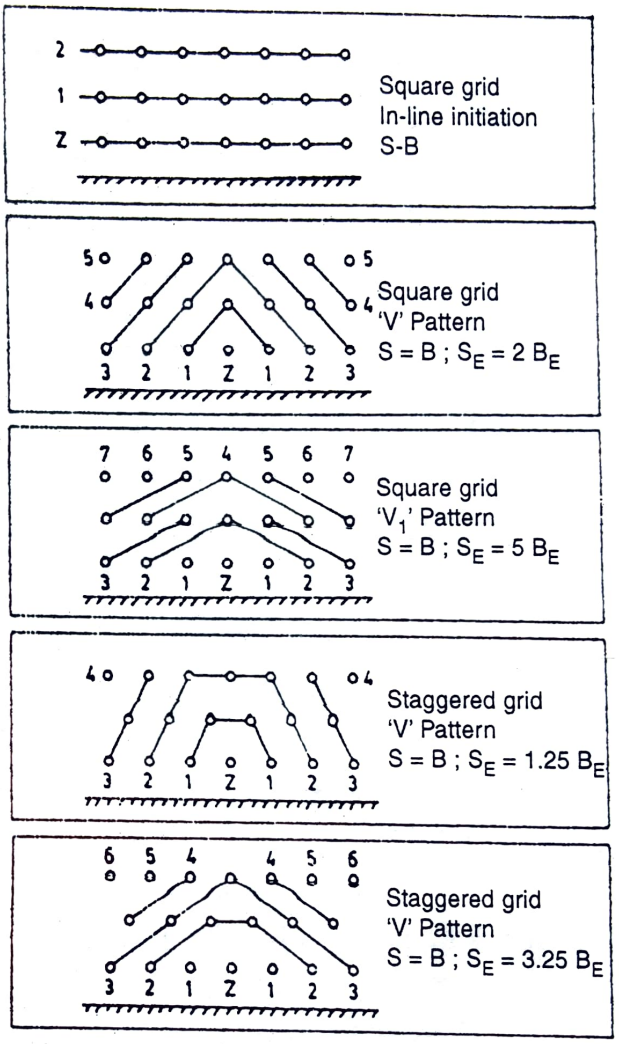


Fig. 32 Multi Row Firing Pattern

The use of delays in between the blast-holes or between the decks in the blast-hole shall indicate the fragmentation to be obtained, noise and vibration to be produced, the direction of the movement of the blasted rock mass, powder factor to be achieved, the condition of new free faces, elimination of toe problems etc.

Flat face delay pattern : It is clearly explained in the fig. 29 and 32. In this fig. 29 the number against each hole shows the order of detonation by the ms delay dotonator and the arrows show the direction of movement of muck piles. This type of delay pattern are not very much effective for achieving the good fragmentation. This type of pattern can be used with the square, rectangular or staggered drill pattern.

The 'V' pattern : The various types of 'V' pattern are shown in the fig. 30 and 32. This type of patterns are useful for any type of rocks and may be used with the square, rectangular or staggered hole pattern. However rectangular or square grids are more common. In case of

rectangular grid spacing is more compared to the burden.

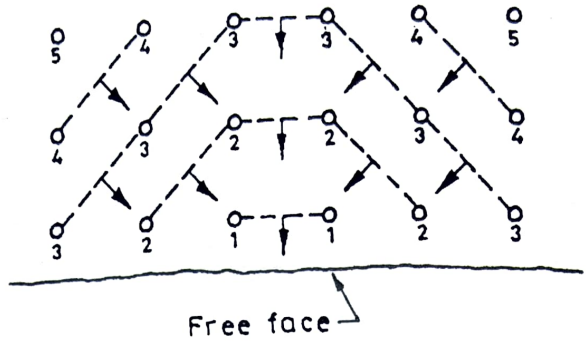


Fig. 30 'V' Pattern

The **ECHELON Pattern :** Whenever a 2nd free face is available in the outside corner and when better fragmentation and displacement are required then this pattern is introduced (Fig. 31).

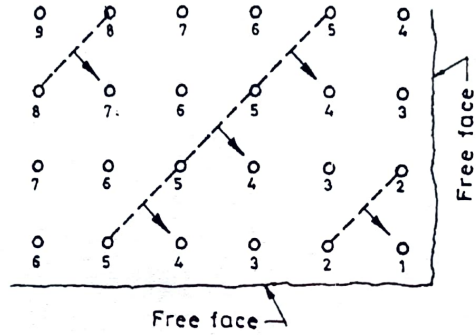


Fig. 31 Echelon Pattern

Question 95 : Discuss briefly about deck charging. What factors are generally considered during the choice among the single row or multi row firing system? What advantages are generally obtained by the short delay blasting? Discuss single row and multi row firing system.

What advantages and disadvantages are associated with the short delay firing with a cut ?

Answer : Deck charging will yield less oversized lump and also it will have smaller uncontrolled zone of rock breaking, Generally in the lower part of the hole 2/3rd of the charge of high strength and density explosive is placed and the remaining 1/3rd charge may be divided into one, two or three supplementary charges in the middle and in the upper parts of the holes. The explosive for the supplementary charges may be low strength and low density type. The number of decks and the length of the air gaps which are left all along the length of the holes depends upon the minimum permissible length of stemming (which may be reduced by 15 to 25% in this case). The length of the air gap between the deck may be

equal to the 20% to 35% of the length of the charge or distance between the deck charge is filled with stemming material of stemming length 60 to 80% of the length of the charge. The smaller length of the air gap is used for blasting stronger rocks. Here initiation is done not with the charge but by an intermediate detonator with the help of a detonator cord.

Decision to choice among the single row or multiple row drilling and blasting depends upon mainly on firing sequence, quality of fragmentation desired, shape of the muck pile wanted and factors of working on the bench toe. Short delay blasting will give better fragmentation, produce less lump, decrease the consumption of explosive and reduce the width of fragmented muck pile. Single row with short delay firing generally implemented in small mines with narrow working berms of benches and also to reduce the overcrushing of the products. This type of firing generally yields more oversized boulders and less amount of yield compare to the multirow blasting with short delays. Besides these multi row short delay firing improves quality of fragmentation, lower down the number of holes to be blasted, make easy to working in a large reserve block, provide opportunity for concentration of work and make easier for scheduling of complete drilling, blasting and loading operation. In single row firing the following patterns may be used.

a) The alternate delay pattern. It is used for soft rock formation.
b) Consecutive short delay pattern. It is used for rock with medium hardness and
c) Short delay firing with a cut is applied for hard rock formation.

The pattern of short delay firing with a cut (made by drilling a row of cut holes at specified distance from the face. The cut holes are overdrilled by 1 to 1.5m as compared to the other holes), has the following advantages :

a) It will provide additional free faces
b) Assist to break the rock boulders by collision among themselves
c) Assist to form the muck pile in a particular direction
d) Ensure good fragmentation

But the demerits associated with it like more amount of back break, difficult to work in toe, more ground vibration, etc. are worth to mention.

Beside cut pattern other pattern of blasting in the multirow of firing are as following :

Transverse cut pattern : They are used where smaller width of muck pile is desired [Fig. 33(i)].

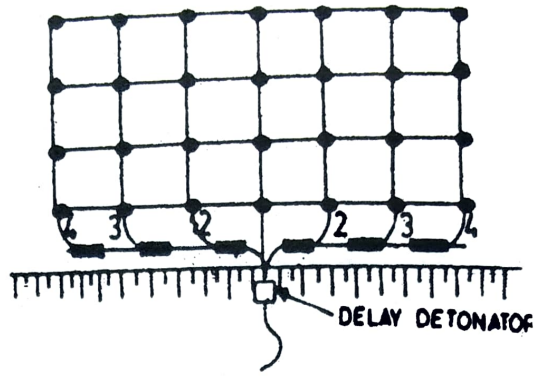


Fig. 33 (i) Transverse Cut Pattern

Wedge or trapezoidal blasting pattern : They are used when the rocks are medium hard and hard one. Due to the motion in the opposite direction in this case, the big boulders are broken by supplementary collision [Fig. 33(ii)].

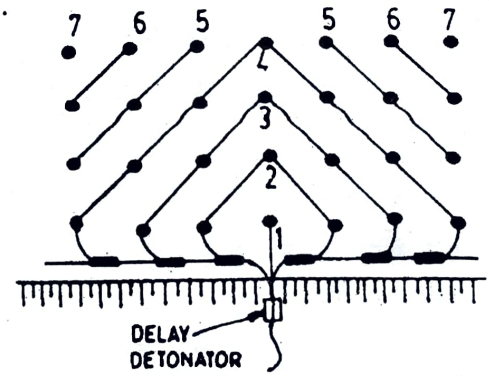


Fig. 33 (ii) Wedge Blasting Pattern

Diagonal blasting pattern : With this type of blasting pattern it is possible to blast the rock towards the least resistance and improve fragmentation of rock [Fig. 33(iii)].

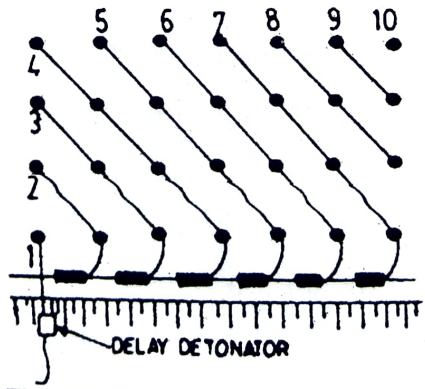


Fig. 33 (iii) Diagonal Blasting Pattern

Question 96 : Discuss elaborately the firing system with the detonating cord in the open cast mines.

Answer :

Firing system with the Detonating cord

- First of all choose the right type detonating cord with the primer.
- Choose delays. There are 3 ways to initiate delay detonating cord

  (1) With a ms connector

  (2) With a non-electric delay detonator

  (3) With an electric delay detonator.

In case of electric delay detonator attach the latter with the downlines (detonating cord) of individual blastholes or trunk lines connecting rows of holes.

However in case of non-electric delay detonator, the latter shall be attached with the detonating cord near to the collar of the blasthole or be used as a surface delay detonator.

The ms delay connectors are jointed into the trunk-line between the boreholes or groups of boreholes.

Inhole delay detonators (non-electric type) may be connected to a cast-booster or be inserted into the primers.

However instead of delay detonators, detonating cords may be directly used for firing the primer. In such case attach the detonating cord directly to the primer.

In case of detonating cord to be used with the cast booster the former to be inserted into the tunnel of the primer, and a knot has to be made at the end so that cast booster remain intact over the knot.

In case of primer is to be connected directly with the detonating cord, one end of the latter is to be inserted into the primer explosive after punching a hole into it. Thereafter the detonating cord and the cartridge are to be taped.

In case of slurry primer, punch a hole into the slurry explosive cartridge about 1/3rd of the distance from the end of the bag, push the detonating cord through the hole and tie the detonating cord thereafter in a tight knot above the end of the bag.

Tie up downlines to trunklines either with the clove-hitch or with the double-wrap half-hitched knots. Tie in downlines as per blast design till all lines are connected.

If the electric delay detonators are used then the detonators, leg wires and connecting lines should replace the trunklines. When the detonators are to be used for delaying rows of holes, the formers are taped with the trunkline of the row for delaying purpose.

Non electric detonators may also be used instead of a detonating cord trunkline.

Use trunkline delays to connect two holes. The plastic sleeve end of the trunkline delay is attached to the downline of the first hole, and the connector block end of the same trunkline delay is attached to the downline of the 2nd hole.

Fire the shot. Firing can be done in two ways-

(a) Firing with detonator and safety fuse

A detonator is crimped with a length of safety fuse (more than 1m length) and thereafter is tapped with the detonating cord pointing front side of the detonator towards the direction of the detonation waves move. Use a hot wire fuse lighter to light the fuse which ultimately fire the shots.

(b) Firing with electric detonator

An electric detonator to be tapped with the detonating cord pointing the datonator towards the direction of movement of the detonation waves. The detonator to be tapped near to the point of initiation of detonating cord. Connent the lead lines with the leg wires of the detonator and test the load line with the blaster galvanometer/ohmmeter/ multimeter after unshunting the lead lines. Connect the leadlines with the exploder/blasting machine and fire the shots.

Question 97 : How is cast booster primer prepared?

Answer :

In the cast boosters there is the presence of two holes in them. One stretched completely from face to face called the 'tunnel' and the other one goes part way through which is termed as 'well'.

For making the primer push the detonator completely through the tunnel and take out the other side of the primer. Thereafter push the

detonator all the way into the well. Tighten up the slack leg-wires and place the primer/booster in the direction of movement of the detonation waves. (Fig. 34)

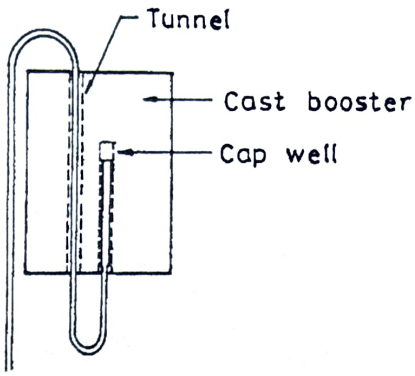


Fig. 34 Making up a cast primer

Question 98 : Design a large scale blast on dragline benches.

Answer : Design of a large scale blast on dragline benches is given in the Table 25.

Question 99 : Enumerate the blast design values in the surface mining operation.

Answer : The Blast design values are enumerated in the Table - 24.

Table 24

BLAST DESIGN VALUES IN SURFACE MINING OPERATION

Parameters	Rock Type					Comment
	Very poor strength	Poor strength	Medium strength	High strength	Very high strength	
RMR	0-20	21-40	41-60	61-80	81-100	
Burden (B) (m)						B = Burden in metre
Shovel Benches	51 x D	44 x D	37 x D	29 x D	23 x D	D = Diameter of blasthole in mm
Dragline Benches	51 x D	49 x D	47 x D	45 x D	43 x D	S = Spacing in m.
Spacing (S) (m)						Stem = Stemming height in m.
Shovel Benches	1.45 x B	1.34 x B	1.25 x B	1.19 x B	1.14 x B	Powder factor expressed in Kg of explosive/m^3 excavated rock
Dragline Benches	1.46 x B	1.43 x B	1.40 x B	1.37 x B	1.34 x B	
Stemming (Stem)	1 x B	0.95 x B	0.9 x B	0.8 x B	0.7 x B	
Powder Factor						
Shovel Benches						
For smaller dia.hole	0.36	0.49	0.61	0.75	0.82	
For larger dia.hole	0.41	0.41	0.51	0.74	0.83	
Dragline Benches	0.27	0.31-0.33	0.34-0.35	0.36-0.37	0.37	

Table 25

	Particulars	Case No. 1	Case No. 2	Case No.3
1.	Rock Condition	Medium hard sandstone (Dry rock)	Medium hard sandstone (Dry rock)	Medium hard sandstone (Dry rock)
2.	Explosive characteristics	Aluminised cartridge slurry and aluminised bulk slurry.	Aluminised slurry and emulsion	Aluminised slurry. Nitroglycerine emulsion
3.	Bench parameters :			
	(a) Bench height (m)	29 to 34	34 to 38	27 to35
	(b) Bench width (m)	70	55 to 70	80
	(c) Length of blast (m)	240	300	400
	(d) Toe burden	12 to 15	12 to 15	12 to 15
	(e) No. of free faces	2	2	2
4.	Drill hole dia (mm)	259	259	259
5.	Burden in 1st row	12 to 15	12 to 15	12 to 15
6.	Spacing for 1st row (m)	5.5	5.5	5.5
7.	Burden for subsequent rows	7 to 9	7 to 9	7 to 9
8.	Burden for last row (m)	9.0	9.0	9.0
9.	Stemming height (1st row) (m)	7 to 9	7 to 9	7 to 9
10.	Stemming height in subsequent rows (m)	6.0	6.5	6.5
11.	Stemming height in last row	9.0	9.0	9.0
12.	Charge length (m)	23 to 28	27 to 32	20 to 23
13.	Average charge per hole (kg)	1400 to 1600	1500 to 1820	1600 to 1800
14.	Charge factor (m^3/kg)	1.54	1.58	1.44
15.	Explosive quantity (te)	382	419	661
16.	Load per delay (t)	22	40	40
17.	Delay interval between rows	75 m.s.	75 m.s.	75 m.s.
18.	Duration of blast (seconds)	1.3	1.4	1.75
19.	Drilling pattern	Staggered	Staggered	Steggered
20.	Firing pattern	In line	In line	In line
21.	Blast results	Very good blast. Some back breaks	Very good blast. Some back breaks	Very good blast. Some back breaks
22.	Peak Particle velocity measured	Not measured	Not measured	45 mm/sec at 1.2 km on free face side 114 mm/sec at 800 m on backside of blast

Table 25 (continued)

	Particulars	Case No. 4	Case No. 5	Case No.6	Case No 7
1.	Rock	Medium hard sandstone (12 to 15 m water column)	Medium hard sandstore (12 to 15 m water column)	Medium hard sandstone dry in front portion, 8 to15 m water column at rear	Medium hard sandstone (4 to 5 m water column)
2.	Explosive characteristics	Aluminised cartridged slurry and emulsion with nitroglycerine explosives	Aluminised cartridged slurry and emulsion with nitroglycerine explosives	Aluminised bulk slurry and cartridged slurry	Heavy ANFO with aluminised primer cartridge and also aluminised cartridge slurry
3.	Bench parameters :				
	(a) Bench height (m)	34.5	34.0	34.0	34.0
	(b) Bench width (m)	90.0	90.0	80.0	80.0
	(c) Length of blast (m)	300	160	193	270
	(d) Toe burden	10 to 12	14 to 16	10 to 12 in some portion, remaining free face blocked by blasted muck	12 to 15
	(e) No. of free faces	2	1	1/2	1
4.	Drill hole dia (mm)	259 to 311	311 to 259	259	259
5.	Burden in 1sr row (m)	10 to 12	14 to 16	as in 3 (d)	12 to 15
6.	Spacing for 1st row (m)	7	7	5.5	5.5
7.	Burden for subsequent rows	8 to 11	8 to 11	8 to 9	8 to 9
8.	Burden for last row (m)	9.0	11.0	9.0	9.0
9.	Stemming height (1st rows (m)	8 and 14 alternatively	8 to 14 alternatively	7 to 8 alternatively	8 and 14 alternatively
10.	Stemming height in last subsequent rows (m)	7.0	7.0	6 to 6.5	6.5
11.	Stemming height in last row	9.0 (decoupled and deck charging)	9.0 (decoupled and deck charging)	6.5 (deck charging)	6 to 6.5
12.	Charge length (m)	25 to 27	25 to 27	27 to 28	27 to 28
13.	Average charge/hole (kg)	1850 to 2600	2600	1700 to 1800	1700 to 1800

Table 25 (continued)

	Particulars	Case No. 4	Case No. 5	Case No.6	Case No 7
14.	Charge factor (m3/kg)	1.58	1.5	1.85	1.55
15.	Explosive quantity (te)	680	342	274	461
16.	Load per delay (t)	25	11	15	16.5
17.	Delay interval between rows	100 m.s.	100 m.s.	100 m.s.	100 to 125 m.s.
18.	Duration of blast (seconds)	2.0	1.625	2.35	1.8 sec.
19.	Drilling pattern	Staggered	Staggered	Staggered in front, square at the rear of block	Staggered and aquare
20.	Firing pattern	In line	In line	V pattern	In line
21.	Blast results	Very good fragmentation 27% sidecast	Very good fragmentation	Very good fragmentation with 8 to 9% throw.	Very good fragmentation. Approx. 20 to 25% sidecast.
22.	Peak particle velocity measured	Not measured	5.5 mm/sec. at 4.5 km distance from blast site and a wave frequency of 10 to 12 hz	3.5 mm/sec. at a distance of 6.1 km from blast site and a wave frequency of 8 to 10 hz	7.56 mm/sec. at 2.34 km distance and air overpressure of 98 db at 20 MVA S/S 3.25 mm/sec. at 6 km distance and a wave frequency of 4 to 40 Hz with mazimum frequencies between, 8 to 10 Hz at singrauli hospital

Question 100 : How are free faces formed in the opencast blasting face? How they effect in blasting efficiency?

Answer : The control of blasting fragmentation is done by the control of consumption of explosives and creation and maintenance of free faces from which the tensile wave reflects after firing the charge. These free faces contribute much in fragmentation and may be formed by the pre-designing of the face after extraction of fragmented rock and also in the process of delay firing. Even by proper designing creation of five to six free faces are possible. The more the number of free faces more the volume of fragmented rock will be produced by overcoming the resistance to shearing and tension of rock.

The use of delays between the bore holes or between decks in the boreholes can determine the fragmentation, flyrock, noise, vibration, the direction of overburden movement, powder factor, condition of the new free face, condition of toe, etc. The delay between the rows must be enough to have a change the fragmented rock to move away and to create a relief (the distance the explosives energy has to travel to get to open area) before the next row detonates. Rock usually takes 60 ms to move around 1 m.

Question 101 : What is fracture dilation?

Answer :

Fracture dilation

The effect of fracture dilation on the stability of the exposed faces will depend on the orientation of the fractures relative to the exposure, the roughness of the fracture surfaces, density of charge in the blasthole, degree of confinement and fixation, etc. Under a high degree of fixation, large diameter charges in the back rows of blast pattern can produce extensive catering, creating a zone of vertical heave which can extend backwords at a distance greater than the height of the bench.

Question 102 : What are the main parameters of blasting which command the yield of brocken rock'?

Answer : Depth of hole, spacing, burden, diameter and inclination of holes are the main parameters of blast hole design. Beside these, type and density of the explosives, the dimension of the drilling pattern on a bench and the blasting sequence mainly command the yield of broken rock.

Question 103 : What is the rock? How fly rock is produced in the surface miner? What are then remedial measures?

Answer : Fly rock is the excessive movement of blasted rock debris at random from the blast-zone which propels through air just after the firing shots. The fly rock is produced due to their tearing off by gases which is ejected out through the cracks at high velocity due to venting of explosive energy violently into the atmosphere in the form of gas expansion energy. More than 50% of blasting accidents in the surface mining operation are caused by flyrock and other projectiles. The Directorate General of Mines Safety has recommended 500 m radius as danger zone for safety in opencast mine blasting.

The following is the popular equation for the projective velocity—

$$V = 1.14\left[\frac{B}{(E)^{0.33}}\right]^{-1.17}$$

$$E = 0.07853 \times d \times \rho_0 \times W_s$$

where V = projectile velocity, m/sec

B = burden, m

E = energy, K.Cal/m

d = charge diameter, cm

ρ_0 = density of explosive, gm/c.c.

W_s = absolute weight strength, cal/gm

$$B_v = 3927.4\left[\frac{St}{d}\right]^{-1.902}$$

where B_v = Burden velocity, ft/sec.

St = Stemming height, ft.

d = Diameter of shothole, ft.

Causes and Remedial measures

- Presence of faults, weak planes fissile cracks, bedding planes, open fractures and joints, mud seams, etc. in the blast zone assist venting of gases through them (because of least resistance) and causes more flyrocks.

  Solution : Inspect thoroughly the blast site for overhang, fractures, open-joints, weakplanes, mud seams, etc. and if they are found adjust the blast pattern accordingly. Avoid drilling

near to these weak planes, fractures, joints, etc.

- Blastholes intersecting solution cavities, voids, etc. produce more flyrock due to heavy concentration of explosive provided that non-cartridge type explosives are used.

  Solution : Backfill in the open fractures, collapsed blastholes, voids, cavities, etc. and avoid drilling near to these places.

- Presence of water in a blasthole may cause floating of explosive charge over the former, increase of specific charge, reduction of stemming column, etc. These factors combinely responsible for yielding of more flyrock.

  Solution : Pump out water from the blasthole before charging. Frequently measure the charge height, stemming height, etc. Use high density slurry explosives and also sand as stemming material.

- If there are the presence of loose stone over the surface of bench, then just after firing shots they eject out very easily when high pressure gases vent through the collar of the blasthole.

  Solution : Remove loose stone before charging.

- Presence of fracture over the solid face due to back break/over break, causes reduction of burden and increase of specific charge in front row of the subsequent blast. It causes rapid venting of explosive gases through highly fracture spots created at the crest level and produce flyrock profoundly.

  Solution : Administrate suitable specific charge and select proper stemming length. Select optimum burden. Use deck charging and shock tube detonating system.

- Larger diameter blastholes produce higher amount of gas and hence causes more fly-rocks.

  Solution : Select optimum diameter blastholes.

- Sometimes improper drilling (deviation of holes) of blast-holes leads to small burden and it generates heavy amount of flyrock.

  Solution : Drill blast-holes accurately at the correct angle as per the blast design, and also make the sub-grade drilling.

- If a blast hole is drilled near to an abandoned blasthole the

latter will act as a free face, gases will vent through the weakest point and it will generate flyrocks.

Solution : Stem the abandoned blasthole totally.

- Close spacing of blastholes causes crushing and cratering effects and produces flyrocks.

  Solution : Maintain optimum blast design.

- In case of low height bench and using larger diameter blasthole in it will cause production of flyrocks.

  Solution : Select optimum diameter blasthole under such a situation.

- Excessive burden will lead gases to vent through the collar of the blasthole and ejection of stemming gives rise to flyrock.

  Solution : Maintain optimum blast design and stemming height.

- Use of high specific charge in the blasthole responsible for production of flyrock.

  Solution : Use optimum specific charge.

- Use of free flowing slurries concentrates charge in the bottom of blasthole may cause production of flyrock.

  Solution : Use low density explosives. Administrate the optimum amount of explosive charge.

- Inadequate stemming will cause explosion gases come out the top of the blasthole at a vary high velocity causing formation of crater and flyrock.

  Solution : Maintain sufficient stemming column. Stemming height must be more than 0.6 times the burden width.

- Use of improper stemming materials does not have interlocking characteristics which lead to generation of flyrock.

  Solution : Use drill cuttings as a stemming material. Crushed rock of 4 mm to 9 mm size may also be used.

- Simultaneous firing and use of ms delay detoantor of less than 17 ms delay produces more flyrock.

  Solution : Use delay blasting with short delay detonator. In multiple shot the delay between the row must be long enough to allow rock from an earlier row to move adequately to give a

relief for movement of burden of the next row of blastholes. Adopt 'V'pattern initiation with adequate inter row delay intervals.

- Secondary blasting causes flyrock.

  Solution : Avoid secondary blasting. Use hydraulic hammer for breaking boulders.

- If front row fails to blast properly the back row holes will creates and generate flyrock.

  Solution : Maintain optimum blast design.

- Collar initiation system - causes more flyrock compare to the bottom initiation system.

  Remedial measure : Use bottom initiation system.

- During dealing misfire holes, flyrock produces due to less burden and spacing.

  Remedial measure : Leave mass of blasted rock in front of the face from the original blast.

- Loading of charge near to the top of blasthole and also careless drilling, blasting and blast management leads to generation of flyrocks.

  Remedial measure : Avoid overloading of explosive charges into the blasthole, drill blastholes accurately at the correct desired angle, do muffle blasting, etc.

Question 104 : Why is the sub-grade drilling done?

Answer : The length of the holes is calculated on the basis of height of the bench, inclination of hole with the horizontal and amount of the subgrade drilling, The subgrade drilling of a blast hole will result better fragmentation of rock at the toe of the bench, But the length of subgrade drilling should be optimum to reduce the volume of drilling work and also to protect the berm of the lower benches.

Question 105 : What are the main demerits of concentration of charge in a blast hole? Discuss briefly about the deck charging.

Answer : To increase the rock fragmentation the explosive charge column should be distributed properly all along the length of the hole. Unnecessary concentration of charge in a blast hole or larger diameter blast hole results in increased production of oversized lump and the

volume of over-crushed rock around the charged hole. The diameter of the blast hole is generally chosen on the basis of production requirement and cost consideration.

Question 106 : What are the main purposes of stemming?

Answer : Length of stemming should be sufficient to prevent escaping of blasting ingredients, outburst of rock and the formation of shock wave. At the same time distribution of charge near the mouth of the hole is to be considered.

Stemming helps production of confine gases and the explosive energy. The best stemming material is coarse gravel because it locks together when upword pressure from the explosion occurs. Wet materials are not good stemming materials because they may slide out of the hole causing a blownout. Drill cuttings are the best stemming materials. Don't use any stemming materials which can damage the detonating cord, blasting wires, etc.

Question 107 :

What are the main advantages which are offered by a bulk system? Design deep hole blasting showing various parameters in

a) Shovel benches

b) Dragline benches

Answer : Bulk systems have a number of advantages and, as already highlighted these include features such as complete boreholes coupling, energy variation to suit strata conditions, density control and the choice of a variety of explosive types such as emulsions, slurries, ANFO, HANFO and doped products. However, ultimate benefits of these advantages are only derived after complete match of equipment and field parameters. Some typical range of parameters are shown Table 26 although variations from site to site are inevitable.

Table 26

EQUIPMENT	**SHOVEL**				**DRAGLINE**					
Explosive	**Strong**		**Medium strength**		**Strong**			**Medium strength**		
Bench height (m)	18	22	18	22	22	28	33	22	28	33
Hole depth (m)	18	22	18	22	22	28	33	22	28	33
Burden (m)	7	9	6.5	8	9	11	13	8.5	10	11.5

EQUIPMENT	SHOVEL				DRAGLINE					
Explosive	Strong		Medium strength		Strong			Medium strength		
Spacing (m)	8.5	10.5	8	9.5	10	12	14.5	9.5	11	12.5
Stemming (m)	7	9	6.5	8	7	9	10.5	6	8	9
Vol./hole (m^3)	1071	2079	936	1672	1980	3696	6221	1777	3080	4744
Hole dia. (mm)	200	250	200	250	250	269	311	250	269	311
Charge length (m)	11	13	11.5	14	15	19	22.5	16	20	24
Charge (kg)	415	766	397	756	757	1403	2222	942	1364	2188
Charge factor (m^3/kg)	2.6	2.7	2.4	2.2	2.1	2.6	2.8	1.9	2.3	2.2

Question 108 : Discuss briefly about the chamber blasting.

Answer :

Chamber Blasting

Generally this type of blasting is practised in the very high benches in soft rock formation, to reduce the total meterage of drill holes spaced at sufficiently large distances. Since maximum charge is required at the toe of the benches,. the size at the bottom of the hole is enlarged by firing one or two small chamber charges. Then total block is blasted like normal blasting method after placing the charges in all the holes made in the ore block. This type of chamber blasting may also be practised in the very hard rock formation where holes are made by Jet-piercing and bottom of the hole is reamed with special type of drill bits.

Question 109 : What is Coyote Blasting, Cushion Blasting?

Answer :

Coyote Blasting

In this system a heavy amount of explosive charge in the order of hundreds of tonnes are packed in the large one or more coyote chambers made by driving tunnels, drifts or raises, to excavate a huge amount of rock in the order of thousands of tonnes from a hilly terrain or rock body where sufficient out of free faces are available for the movement of dislodged rock. After packing explosive in the Coyote Chambers, the connection tunnels/drifts/raises near the chambers are backfilled tightly with the excavated muckpiles and the charge is normally blasted with the help of detonating cord.

Cushion Blasting

In this system a heavy amount of closely spaced and burdened (same dia. as the production hole) vertical holes are drilled all along the periphery (beyond the last rows of holes in the bench) of the blasting zone in the bench. They are loaded with light explosive charge and nicely stemmed and are blasted after the blasting of the main charge with the help of delay detonator. Deck charging may also be practised in this type of blasting. Generally the diameter of the explosive cartidges are half the diameter of the blasthole. The cartridges are generally lowered into the hole by attaching with the detonating cord at predetermined intervals. The remaining air space in the hole is completely back filled with the stemming material.

The blast holes are fired after the rest of the round has been blasted. Generally, nitro-glycerine based cartridged explosives are used for this controlled blasting technique. The diameter of this cartridges are less than 50% of the diameter of the blasthole. The cartridges are generally spaced by a string although in some of the cases they may form a continuous column. Higher density expllosive is put at the bottom of the hole. The diameter of the hole may vary from 37.5mm to 150mm, the diameter of the cartridge vary from 22mm to 50mm (depending upon the size, spacing and burden of the blast holes), spacing of charges varies from 100mm to 550mm, approximate weight of charge varies from 0.14 Kg/m to 1.47 Kg/m length of the blasthole, spacing of boreholes varies from 0.6 to 2.1m, burden of boreholes varies from 0.9 to 2.7m, etc.

All the holes meant for cushion blasting are fired simultaneously using a detonating fuse trunk line. In some of the unconsolidated rock formations upper half of the holes are lightly charged to prevent back breaking.

Question 110 : Why is secondary blasting practised? Discuss the following.

(a) Pop shooting,

(b) Plaster shooting.

What advantages are offered by plaster shooting?

Answer : Secondary blasting is done to break oversize boulders produced during the primary blast, to a size suitable for thc crushing plant. There are two basic methods of secondary blasting : Pop shooting and Plaster shooting.

Pop Shooting

The boulders are to be selected first and shall be marked for drilling holes. The number of holes to be drilled depends upon the size, shape, type and placement of rock boulders. The depth of hole shall be around 2/3rd to 3/4th size of the rock. Insert cap sensitive cartridge explosive into it (generally 200 gm of explosive for every cubic metre of the boulder) and fire after taking due precautionary measure (Fig. 35).

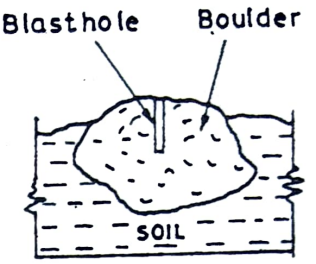


Fig. 35 Pop Shooting

For pop shooting, a hole is normally drilled just beyond the centre of the boulder to be broken. With some kinds of rocks, however, shotholes 30 cm deep are sufficient to break every large boulder.

The charge varies with the size of the boulder but, for average conditions, a boulder 90 cm x 60 cm x 60 cm requires a charge of about 130 g of Special Gelatine–80% strength. The shots can be fired using either safety fuse and plain detonators or electric detonators.

If electric shotfiring is practised the electric detonators are connected in series and the shots are fired simultaneously. It is necessary to withdraw mechanical loading equipment to a safe distance, since there is considerable scattering of rocks when firing pop shots.

- Number of holes per bulder : 1 or 2 depending upon the size.
- Depth of hole : 2/3rd of the thickness of boulder. Minimum depth 0.5m.
- Charge per hole : 60gm to 100 gm of special gelatine (Depending upon the boulder size)
- Dia of blast-hole : 35mm
- Initiation device : Ordinary electric detonators
- Muffling procedure : Sand bag covering over each hole
- Expected volume of rock/blst-hole : $2m^3$

Plaster Shooting

Plaster shooting provides a ready means of breaking even large boulders in circumstances where drilling is difficult or expensive. A high-velocity, high-strength gelatine type explosive is most suitable for this work. A charge of one or more cartridges/slabs is primed with a detonator and safety fuse, or an electric detonator, and laid on the surface

of the boulder. It is then covered with a shovelful of plastic clay, which is pressed into position by hand. It is advantageous to wet the surface of the stone before plastering, and the clay should be well pressed down so that it is in good contact with the surface of stone around the explosive.

In plaster shooting, the charges used are about four times those required for pop shooting.

Advantages of Plaster Shooting :

1. No drilling is required, thus saving labour and compressed air, a consideration of particular importance in hard and abrasive rocks which may be difficult to drill. For the same reason, even though the amount of explosive used is much greater than in pop shooting, it is found to be more economical in overall cost.
2. A group of plaster shots can be prepared far more quickly than the same number of pop shots.
3. Boulders are broken where they stand and the fragmented material is not scattered over a wide area.
4. There is less likelihood of damage through flying debris. This is advantageous since equipment need not to be moved far to a place of safety and thus less loading time is wasted when secondary shots are fired.

Question 111 : How is a blasting pattern designed in a surface mine considering the employment of existing equipments?

Answer : Before designing a blasting pattern the capacity of existing mining equipments like shovel, dragline, dumpers or other transporting method, crushing plant, drills, blasting machine/exploder etc. are to be ascertained. Then blasting site, muck piling area, locality, important structures near to the vicinity, etc. are to be inspected carefully. At the same time availability of the types of explosives, detonators and other blasting accessories, safety guidelines imposed by the DGMS etc. should be considered into great depth. First of all depending upon the capacity of the dragline, shovel, transporting media, crushing plant etc. find out the optimum size of the desired fragmented rock. Based on these results and equipment capacity find out the height of the bench, spacing and burden of the holes, diameter of the holes, etc. Here factors like casting of muckpile, ground vibration, safety, etc. are also to be looked into.

After selecting the bench height, spacing and burden of the holes, select the length of the hole (considering inclination, if any, length of the toe, etc.) and depending upon the locality and presence of important structures near to the vicinity select the amount of explosive charge/hole, delay interval, firing sequence, number of rounds, etc. During that also consider the factors like ground vibration, availability of delay detonators and exploder, size of fragmentation, angle of breakage and desired direction and place of throwing the muckpile, etc. But before everything all the safety measures which will be imposed by the DGMS like control blasting, muffling, marking of danger zone, evacuation of men and equipments, maximum permissible charge per blast hole, number of charge per round, etc, should be given paramount importance. For minimising the ground vibration lower down the distance of spacing and burden between the holes and restrict the amount of the maximum charge accordingly. After establishing the geotechnical parameters of the rock if optimum charge is calculated correctly and also short delay detonators are used, then ground vibration, throw of the muckpiles, air blast, noise, etc. can nicely be controlled.

Question 112 : What is satellite charging?

Answer : It is the secondary charging of explosives by the side of the main explosive charges. Smaller diameter blast holes are drilled by the side of the main blast holes, at a very low depth (2-3 m depending upon the site - specific conditions) and thereafter lightly loaded with 3 to 5 kg of smaller dia cartridge explosives. Thereafter these satellite blastholes are fired along with the main blast. The satellite charging will help in not to form any big boulders from the mouth of the main blast and will also help to tackle the cracks created from the previous blast.

Question 113 : Write short notes on the following :

a) Axial Priming

b) Multiple-point Priming

Answer :

Axial Priming

Here a central core of high explosive primer is placed throughout the length of the ANFO or slurry explosive column. Depending upon the diameter of the blast-hole, the central core may be a high strength detonating cord or cartridge of explosives usually of presplit type, butted

end to end. In reality the primer will impart side initiation to the ANFO which propagate across the borehole diameter.

Advantages :

i) If the central core consists of a very high velocity explosive the reaction within the borehole will be completed within a shorter time compared to the end priming.

ii) There will be a less formation of incompetent rock since it will reduce loss of confinement casused by premature burden movement.

iii) The higher bulk strength of the primer as compared to ANFO or slurry explosive is also advantageous but not commensurate with the additional cost involved.

Disadvantages :

i) More time and expense is required to put the high explosive core into the blasthole.

ii) Since it presents no significant advantages to outweight its higher cost, the axial priming system has not come into general use (Fig. 36).

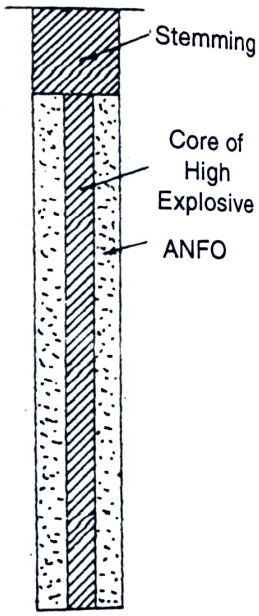


Fig. 36 Shows a schematic diagram of axial priming system

Multiple Point Priming

Here numerous primers are spaced up the blasthole to assure complete propagation of the column charge. When cast primers are strong on a detonating cord downline, the detonation of the cord outruns that of the ANFO and initiates each cast primer slightly ahead of the on coming ANFO detonation front. This results in a series of transient velocities in the ANFO and also a series of collisions of detonation waves. Once a stable detonation has been initiated in a large diameter ANFO column, the detonation will propagate throughout the column at its steady state velocity. So assuming a good ANFO mixture, multimple priming is usually unnecessary. Conditions under which multiple priming can be practised are as follows :

i) Blasting with deck charges

ii) Where slip planes are present in the rock and hazard of charge separation is present.

iii) The existence of two or more difficult breakage zones each requiring maximum explosive confinement for optimum results (Fig. 37).

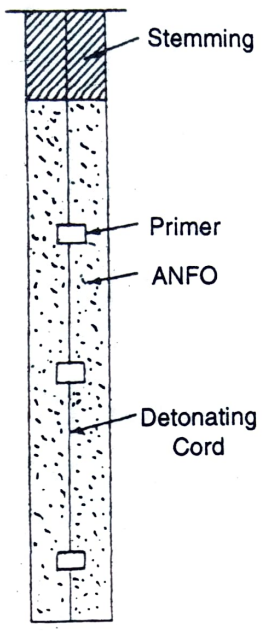


Fig. 37
Shows a schematic diagram of multiple-point priming system

Question 114 :

(a) How is a primer cartridge to be prepared and loaded for blasting in opencast mines? How are cast booster, cartridge primer and water gel primers are made?

(b) How is loading of explosives in the blast holes done in the surface mine? What safely measures are to be taken during loading the explosive charges in to the blast hole?

(c) Discuss briefly about the stemming system in the surface mine.

(d) Before connecting any wire what checks are to be made?

Answer :

(a) Priming

A primer is a high explosive cartridge incorporating a detonating device. In general sense priming is the placing of either electric detonator, ordinary detonator attached to the safety fuse, detonating fuse, etc. in a cartridge of high explosive or placing of detonating fuse threaded into booster like primex, pentolite, etc., inside a bag containing blasting agent or coiling and tieing up of the detonating fuse around thc cartridge like LOX, etc. If OCG is used as primer cartridge then with the help of a brass pricker the explosive cartridge is pricked (approximately 20 cm deep) and one end of the detonator fuse is pushed into it and a knot is made so that the fuse end should not come out of the cartridge. The cartridge is lowered down into the hole with the help of the same detonating fuse.

If aquadyne or toeblast explosives are used then the explosive cartridge is pricked (approximately 20 cm deep) and one end of the detonator fuse is pushed into the hole. The cartridge is then coiled by the same detonating fuse and at the end it is to be tied up tightly with the

help of a knot. The primer cartridge is lowered down into the hole with the help of the same detonating fuse. The primer cartridge is generally placed at the bottom of the hole in reverse detonation to effect the blasting at the bottom of the hole where the burden is the maximum although during deck charging or at the time of long hole blasting more number of primer or booster explosives are placed in the required distance of the explosive column in the hole. Reverse detonation should not be practised for safety fuse firing, since after firing the contact fuse with the charge at the front will lead burning of explosive before detonation. The detonating fuse at the outside of the charged holes are cut in a length of 1.2 to 1.5 m and attached to the trunk line of detonating fuse to blast. In the deck charged blastholes the detonating fuse should be threaded through atleast one cartridge per deck at the bottom of thc charge column in each deck. ANFO, slurry explosives, etc. are generally initiated by a booster charge and the booster charge is initiated by a primer. The percentage of booster charge in case of a high explosive varies from 5% for hole above 150 mm diameter, 10 to 15% for the holes varying from 100 mm to 150 mm hole diameter and 20 to 25% for the holes 50 mm to 100 mm diameter. But in case booster like primer or pentolite, the detonating fuse is threaded through them. Then the threaded booster is packed inside a bag containing the blasting agent to form a primer which is then lowered down into the hole by a nylon cord. This type of primer is generally placed at the bottom of the charge but in longhole blasting additional fuse threaded boosters are to be provided at a predetermined distance along the hole.

In case special ordinary detonator is to be used with the safety fuse, then the end of the safety fuse should be nicely cut, inserted into the detonator and then it is to be crimped at the collar by a suitable crimper. With the help of a brass pricker then one end of the cartridge is to be pricked axially in the centre after opening the wrapper, and the detonator is to be pushed into the hole. The wrapper after folding back should be tied up tightly around the fuse. The detonator should be pointed towards the main charge. Electric detonators are also similarly inserted into the primer and then the lead wires are coiled around the cartridge and tied up.

If LOX type of explosive is used then the detonating fuse is to be coiled around the neck of the LOX and is tied up for enabling a good contact of the fuse with the explosive. LOX is lowered down into the hole with the help of self detaching hook attached with the rope. The

hook is hooked with the wire loop attached with the neck of the LOX and then it is lowered down into the holes and self detaching hook comes out automatically from the wire loop when rope is slacked. In watery hole LOX must be packed in a plastic or polythene bag.

Making of Primer

Cast primer - A cast primer contains two holes (a) one hole which goes all the way through (called the tunnel) and (b) other one goes to half way (called the well). The blasting cap (detonator) inserted through the tunnel and afterward pushed inside the well. The slack legwires are pulled lightly. During putting the primer inside the blasthole ensure that the detonator is oriented properly towards the main charge. (Fig. 34)

Cartridge primer - When the cartridge primers are larger than the 50 mm in diameter two holes are punched into the cartridge with the help of a punch made of non-sparking material. One hole is straight and the other one is in slanted position. Make a loop in the legwires and push the loop into the slant hole keeping the detonator in position in the vertical hole. Then tie up the knot as shown in the fig. 38 :

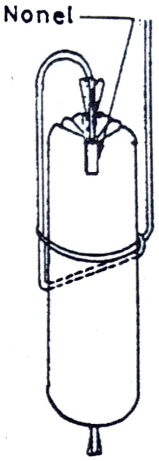


Fig. 38
Making up cartridge primer with nonel

Water-gel primer - Punch a hole in one end of the water-gel cartridge and push the detonator inside of it. Tie half hitches about 1/3rd of the way from each end of the cartridge with the legwires and take up the slack in the legwires. (Fig. 39)

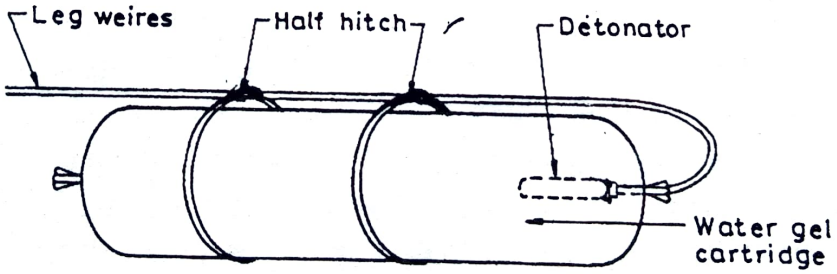


Fig. 39 Making of primer with the detonating cord.

Making of prmer with the detonating cord are shown in the fig. 40 and 41. In the fig. 40 the detonating cord is tied up with the slurry explosive where as in the fig. 41 the detonating cord is inserted into the punched hole are therafter is tied up with a knot.

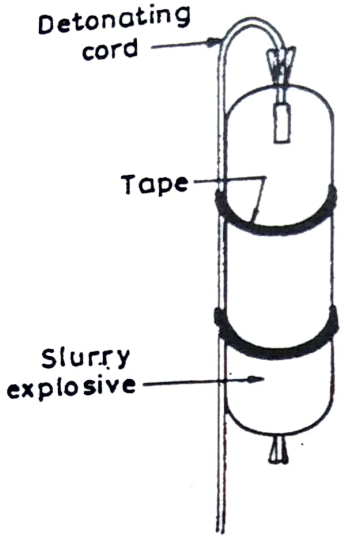


Fig. 40 Priming of slurry explosive with detonating cord

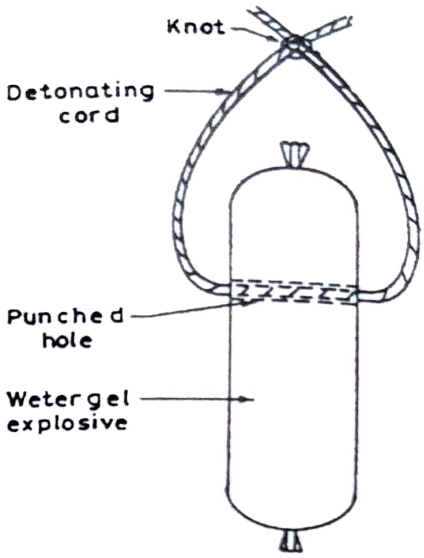


Fig. 41 Priming of water-gel explosive with detonating cord

(b) Loading

Before loading of explosives in the blastholes of the surface mine it should be properly cleaned either by air flushing when the holes are in dry condition or by water flushing in case of moist holes.

When cartridge explosives are used they are selected in smaller diameter (3 mm to 25 mm approx. depending upon the diameter of the blastholes method of drilling, geotechnical parameters of the rock) compared to the blasthole diameter. Loading of primer into the blasthole has been already mentioned in the previous answer. After placing primer in the shothole, cartridge explosives are generally dropped which go down under gravity and produce enough good loading density. However heavy and very sensitive cartridges should not be dropped into the blast hole filled with water or in a case when the side of the blast holes are jagged. But they should be lowered down with the help of nylon cord along with a self-detaching hook made of brass. The cartridges should be suitably taped in steady pressure for squeezing out of the explosive and for filling the hole. The tamping is done with the tamping block in case of long larger diameter blast holes or by pneumatic cartridge loader or by a set of short wooden rods each of which is approximately one metre long and strung on a rope passed through them via axial hole in case of long narrow blastholes.

LOX is lowered into the blasthole in the similar manner as mentioned in the privious answer.

ANFO, slurry explosives, free flowing powder explosives are

poured into the blast holes under gravity and a good loading density can be obtained. Slurry explosives, emulsions, emulsified heavy ANFO, etc. are now a days pumped into the blast hole by the pump trucks. Straight emulsion is charged through a hose pipe, mounted on a hose reel by a pump truck. Boreholes can be bottom loaded where horeholes are dry or where water column is not much in the boreholes. The high density of the emulsion ensures satisfactory sinking and displaces water.

The pump truck also has the ability to deliver Emulsion/AN-FO blends, the emulsion percentage varying between 30-70%. This explosive is delivered through a system of hydraulically driven augers and holes are top-loaded by placing the discharge end of the auger directly above the boreholes.

The pump truck is also provided with Load Cells or RPM based digital meters which accurately record the weight of the explosive delivered into a borehole. Explosive quantities can be pre-set and micro processor control unit accurately stops discharge after the pre-set quantity is delivered into the boreholes.

For long hole/deep hole blasting a very low loading density is desirable which can be obtained by distributing the charge by putting wooden spacers (20 to 50 cm long) in between the cartridges/group of cartridges in case of narrow hole or by distributing the charges in two or more decks separated by stemming in between the charges along the hole.

Deck charging needs a separate primer for each deck, takes more time and also very complicated.

Precaution to be taken during loading of the charges into the borehole

Weather :

- Avoid any electrical storm and lightning, which may cause initiation of detonators.
- Avoid dry conditions, dust storms or snow storms otherwise there may be the danger from static electricity.

Also avoid the zones of stray current which is due to the poorly insulated or improperly earthed electrical equipment, due to radio waves induced from high voltage transmission lines, due to static charges built up in pneumatic loading devices, etc.

Guarding :

Before blasting all persons must be cleared off outside the blasting site beyond a radius of 500 m. Watchmen must be posted with hooters, or loud speaker and red flags in all the vaulnerable paints.

Enough siren must be blown at an ample time before blasting.

Checking boreholes :

Check for cracks, depths, clear passage, watering condition, temperature, etc.

Depth of the hole is measured by a tape attaching a weight at the bottom. In case of presence of water the tape will be a little bit lighter in weight at the touching point of water. Measure this depth (say h).

Hence, drilled depth - h = depth of water

If the borehole depth is less than the drilled depth, it means that the hole is caved in. The caved rocks may be tamped by a pole/heavy object or may be redrilled. In case the length of the drill hole is less than the designed depth, redrilling must be done.

Water : In case there is the presence of water into the borehole, pump out the water, and line the hole with tubing (plastic lining) or use ANFO covered with a plastic bag or use slurry, or emulsion explosives. Tubed ANFO must not be tamped.

Hot holes : To check hot holes use a thermometer lowering down into the hole with a rope and keep it there for about 4 minutes. Thereafter take it out and check the temperature.

In case the temperature in the hole is very high use thermister sensor.

Loading of primers may be done at the bottom of the hole or at the top of main charge or in the middle of the main charge. In case of cartridge type primers, the latters are loaded by the detonating cord/ tubing into the borehole and then it is cut at the top leaving 1 m of the cord/tubing outside and thereafter tail of the latter is wrapped with a stone/wooden stake.

(c) Stemming

Stemming in the shotholes is essential to prevent blown out shots for efficient utilisation of explosive energy and also to control the production of noxious fumes with some special means. Stemming should

be of sufficient in length to offer sufficient confinement to the charge and it depends upon the strength of rock, type of stemming material and the type of explosives. High strength rock and explosives with low VOD require more stemming length. Sand of 0.25 to 2 mm size is the best stemming material if it is properly stemmed with compressed air. In large opencast mines drill cuttings are also used as stemming material, After filling the holes with the stemming material it is tamped with wooden stemming rod.

Before connecting any wires check the following :

- Check the detonators regarding their healthy conditions and resistance. The detonators are checked to know that the legwires are not sorted, nor broken but match with the manufacturer's rated resistance,
- Check each series circuit before connecting them with other series circuits,
- Check the leadline before connecting with the total system,
- Check the exploders and their compatibility,
- After connecting each series with the appropriate connecting wires, find and fix any faults in any of the series circuits and retest, connect the leadline to the rest of the firing circuit, and measure the resistance of the completed firing circuit,
- Check for current leakage using the blasting multimeter,
- If no current leakage is present connect the leadline with the exploder and fire after taking all due precautionary measures.

Question 115 : Discuss elaborately about the blasting practices in open cast mines in India.

Answer :

Blasting Practices in Opencast Mines in India : An Overview

Drilling Method

Rotary and down the hole drills are commonly used for drilling in Indian opencast mines. DTH are compressed air operated at a pressure of 7 kg/cm^2 with hole diameter 100 mm to 225 mm and generally used upto a depth of 12 to 15 m. However, DTH operated at pressure of 14.1 kg/cm^2 with hole diameters of 311 mm to 380 mm may be widely used in Indian Coal Mines in future. Rotary drilling method is used in India for

larger diameter hole and in rocks of very high compressive strength. Generally tricone roller bits in conjunction with high pull down force are employed for satisfactory penetration rate. Drills upto 400 mm dia and a depth of more than 20 m can be practised with the rotary drilling method in hard rock formation.

Bench Height

In opencast coal mines majority of bench height is 6 to 12 m where 4.6 m^3 or 6 m^3 shovels or 8 m^3 front-end loaders are in use and with 10 m^3 capacity shovel bench height is generally 10 to 12 m. In some of the big opencast mines where 24 m^3 bucket capacity dragline with 96 m boom are in use, there bench height is more than 30 m.

Blast Hole Diameter

In opencast coal mines the diametor of blast hole is generally 100, 150 or 163 mm upto a bench height of 8 m whereas beyond 8 m bench height 250 mm blast hole diameter is chosen.

Spacing and Burden

In coal and coal measure rocks the spacing varies vares from 3 to 8 m whereas the burden is selected about 0.8 times thc spacing.

Depth of Sub-grade Drilling

In coal mines the length of sub-grade drilling for a bench height 8 m is generally 0.3 m.

Explosive Used

Nitroglycerine based explosive, slurries, ANFO, and LOX are the main explosives which are used in Indian Coal Mines. Sincc slurries are safe to use, having highwater resistant, high strength and high coupling characteristics to give very good fragfmentation they are now predominantly used in most of the opencast coal mines in India. Presently in India metalized and aluminium slurry are used and also site mixed slurry (SMS) has been introduced. Although the cost of ANFO (94% ammonium nitrate and 6% fuel oil) is very less and easily available, it has several problems to use in the opencast mines. Thc problems are low density, develops too low a pressure, non water resistant and may also cause many accidents due to the use of ANFO (although ammonium nitrate blasting agent with limited water proofness named GN_1 is now a days available). Due to limited availability of oxygen, making of reservoir and soaking plant near the mining site, the

difficulties in preparation of the cartridge and availability of other safer and efficient explosive, the use of LOX is only limited in the few opencast coal mines in India. In coal mine an especial oxygen based explosive LOXITE-C is used. Table-22 mentioned in the Question 67 shows the characteristics of different bulk explosive systems in use in India.

Iron Ore Mines

In Indian Iron ore surface mines majority of the bench heights lie in between 10 to 12 m and blast hole diameter is 150 mm. However, if the bench height is less than 6 m in height, 100 mm diameter blast holes are chosen. In rare cases 250 mm blast hole diameter is selected. In case of 150 mm to 165 mm blast hole diameter the spacing and burden varies from 3 to 7 m whereas the height of the bench varies from 8 to 12 m.

Stemming height	:	30 to 40% of the depth of hole.
Subgrade drilling	:	10 to 20% of the depth of hole.

Where adequate face length is available single row firing system is practised by the use of suitable number of delays. Explosives which used are LOX, opencast gelignite, GN-1, slurry explosives, ANFO, metallized and aluminium slurry, site mixed slurry (SMS), etc.

Limestone Mines

In limestone surface mines in India majority of the bench height lie in between 9 m to 12 m and blast hole diameter is 100 mm. The spacing and burden varies from 3 to 5 m. In many limestone mines in the world 150 mm to 160 mm dia blastholes with a spacing and burden ranging from 5 to 7 m and 4.5 to 6 m respectively are quite in use. Only in very few cases 250 mm dia blasthole are used.

Subgrade drilling	:	10 to 20% of the depth of hole.
Stemming height	:	20 to 40% of depth of hole.
Explosives used	:	OCG, ANFO, GN-1, slurry-explosives, etc.

Question 116 : Discuss about the blasting practices in the following opencast mines:

a) Coal

b) Iron Ore

c) Limestone

Answer : Consult the answer to Question 115.

Question 117 : Give examples of blasting practices in open cast limestone mines and in the overburden benches of a coal mines.

Answer :

Low Cost Blasting Practices in the Limestone Mines

Equipment	:	Shovel : bucket capacity 3.2-3.9m^3 Dumper : Capacity : 35 tonne
Height of bench (m)	:	5-9
Burden (m)	:	2.5
Spacing (m)	:	3.0
Sub-grade drilling length (Percentage of blast-hole)	:	7-10%
Diameter of blasthole (mm)	:	100
Explosive to be used	:	Optimex (It is the low density Ammonium Nitrate Prills manufactured by DEPCL. Average bulk density : 0.8 gm/c.c.)
Percentage in total charge	:	85% to 87%
Kelvex 'p'	:	5 to 7% of the total charge
Low density booster	:	8 to 10% of the total charge
Height of stemming (m)	:	2.5 to 2.8
Number of rows of holes	:	2 to 4
Number of holes/blast	:	50 - 100
Achievable Powder factor (tonne/kg of explosives)	:	6.5 - 7.5

Blasting Practices in Coal Mines

Overburden Bench

Height of bench (m)	:	13-14
Width of bench (m)	:	20-25
Average Burden (m)	:	4
Average Spacing (m)	:	5
Dia. of blast-hole (mm)	:	250

Number of blast-hole/blast	:	20-35
Blasthole pattern	:	Staggered/rectangular
Depth of hole (m)	:	13-14
Length of face (m)	:	40-100
Explosive used	:	Site mixed slurry VOD (m/sec) : 4500 Relative weight strength : 97 (ANFO - 100)
Av. charge/hole (kg)	:	200-220
Charge length (m)	:	4.5
Specific charge (Kg/m^3 of blasted rock) achievable	:	0.55-0.65
Equipment used	:	Shovel - bucket capacity - 4.6m^3 Dumper - capacity - 35 tonne

Question 118 : Discuss blasting practices in surface copper mines.

Answer :

Blasting Practices in Surface Copper Mines

In copper mines the burden and spacing in the blast design are kept around 3.0 and 3.5 and 5 to 5.5 m respectively. If toe problem is high the holes are drilled closer to the bench edges and the spacing of holes are maintained maximum around 5.0 m. The production blast in a clean free face have 3 to 5 rows of staggered blast holes. In the final row the amount of subgrade drilling is around 7 times the dia. of the blast holes. It has been seen that in place of cartridge explosives using strong bulk explosives alongwith 2 primer each initiated by a non-electric delay detonator with nominal delay time of 200 ms will improve blasting efficiency. Charge factor from 0.9 to 1.0 kg/m^3 will reduce to 0.76 to 0.78 kg/m^3 with the use of inclined hole blasting. The diameter of blasthole in a copper mine is 165 mm and 251 mm. The larger diameter hole will reduce the excessive toe problems.

An Example :

Rock type	:	Hard granite
No. of holes	:	46
Bench height	:	13.5 m

Subgrade drilling %	:	8
Depth of hole	:	14.5 m
Burden	:	2.8 m (for larger number of holes) 3.4 m (for fewer number of holes)
Spacing	:	4.6 m (for larger number of holes)
Drilling pattern	:	'V' pattern
Water in holes	:	7 to 8 m
Explosive Combination	:	Indoboost (HE) and Indogel-270
Average charge per hole	:	200 kg
Percentage of secondary blasting	:	2%
Stemming height	:	3 to 3.5 m
Delay pattern	:	I,II,III,IV,V,VI,VII,VIII delay number
Calculated volume before charging	:	9400 m^3
Actual volume	:	12100 m^3
Charge factor (while charging)	:	0.96 kg/m^3
Charge factor (after mucking)	:	0.76 kg/m^3
Explosive cost	:	Rs. 16.28 per m^3
Amount of throw	:	40 to 45 m along the face
Fly rock problem	:	Negligible
Back break	:	4 to 5 m
Fragmentation	:	Excessive and uniform throughout the block
Damage of any machinery	:	NIL

During blasting enough precautionary measures are to be taken to protect mining machineriy, power cables, electrical installations, pumps, etc. by providing proper covering to the same so that the fly rocks can not damage them.

In the surface copper mines the predominantly used explosives are given in the following Table-27.

Table 27

Explosives used in Surface Copper Mines

Manufacturer	Product	Ratio %
IBP Co. Ltd.	Indoboost	30
	Indogel - 250	40
	Indogel - 210	30
Indian Exposives Ltd. (ICI)	OCG	40 - 50
	PG - 2	60 - 50
IDL Chemicals Ltd.	Toe Blast	40 - 50
	Supergel	60 - 50
Naba Bharat Explosive Co.	Bharat Toe	40 - 50
	Bharat Base	60 - 50
Indo Gulf Explosives Ltd.	NCN - 600	40 - 50
	NCN - 500	60 - 50
Maharashtra Explosive Ltd.	Mex blast	40 - 50
	Mexam I & II	60 - 50

Question 119 : Discuss blasting practices in surface limestone mines.

Answer : In most of the limestone mines the height of the benches varies from 4 to 11 m. For benches less than 7 m in height sizes of the blast holes are around 100 mm with a burden and spacing both around 3 m irrespective of the height of the benches. The yield of powder factor is around 5 to 6 tonnes per kg of explosive. However, after taking into account the secondary blasting the powder factor comes down to 3 to 4 tonne/kg of explosives. The blasting of benches with a height lower than 7 m causes problems like hard toe formation, poor blasting and also poor diggability by the shovel in the blasted muck pile.

The following are the examples of blasting practices in the limestone mines :

Case 1 :

Diameter of hole	:	100 mm
No. of holes in each blast	:	60
Hole depth	:	6.25 m

Burden	:	3.05 m
Spacing	:	3.05 m
No. of rows	:	4
Pattern of holes	:	Multiple 'V' cut
Explosive used	:	Charge per delay, OCG - 1000 kg, Indogel - 230 kg, ANFO 1550 kg.
Initiation	:	Both downline and trunk links of cordtex (detonating fuse) and 6 to 10 numbers delay are used.

Charge details

Delay No.	No. of holes	Charge (kg)
6	8	388
7	17	824.5
8	17	679
9	14	679
10	7	339.5

Case 2 :

Height of the bench	:	11 m
Diameter of the hole	:	150 mm
Spacing	:	7.5 m
Burden	:	4.2 m
Depth of hole	:	11.5 m
Subgrade drilling	:	0.5 m
Charge per hole	:	155 kg to 160.03 kg
Total charge	:	5075 kg to 5280 kg
Maximum charge/delay	:	160.03 kg
Explosive used	:	40% toe blast + OCG or only OCG as a booster charge. 60% ANFO as main charge
Stemming height	:	2.5 to 3 m
No. of holes	:	33

Charging method

I. 7 Cartridges of 125 mm dia and 6.25 kg OCG booster at the bottom of the hole.

II. 50 kg prilled ANFO followed by 2 cartridges of OCa and 50 kg ANFO, one OCG cartridge at the top before stemming.

Initiation is done by detonating fuse trunk line laid all along the each row. 25 millisecond cord relays are placed in between each hole alongwith the line of the millisecond delay detonator. Both the down and trunk lines are laid in pairs. Both trunk lines of detonating fuse are tied together all along the length to take care of the discontinuity of one line by the other to eliminate misfire.

Due to heavy blasting with this method the number of blast in a month is around 4 to 5. This practice improved fragmentation, reduce overbreak/back-break, reduce ground vibration, control the height of the muck pile for easy excavation by a 3.5 m^3 hydraulic Shovel, yield smoother face and enhance the powder factor. The capacity of the dumpers used in the mines are 35 tonne.

The blasting pattern and the distribution of various explosives in the blast hole has been shown in the Fig. 42 and Fig. 43 respectively.

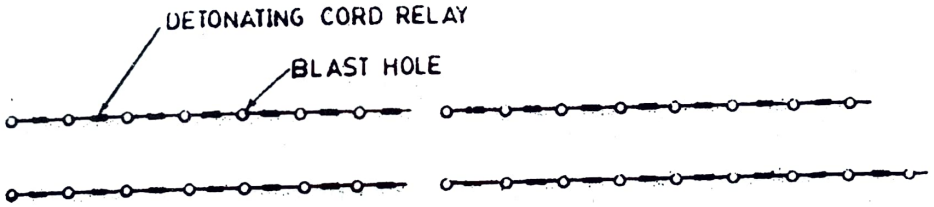


Fig. 42 Shows the blasting pattern

Question 120 : Discuss briefly about the parameters which are affected by blasting performance and also parameters on which a DTH machine depends, What are the main reasons behind a bad blast and difficulties in drilling?

Answer : All operations like shovelling, secondary blasting, grade control, crushing, screening, etc. are affected by blasting performance. Generally 'down-the-holes' (DTH) percussive drilling varying in diameter from 100 mm to 250 mm, operating at 7 kgf/ cm^2 pressure are widely used now a days. The work output of each machine can be known

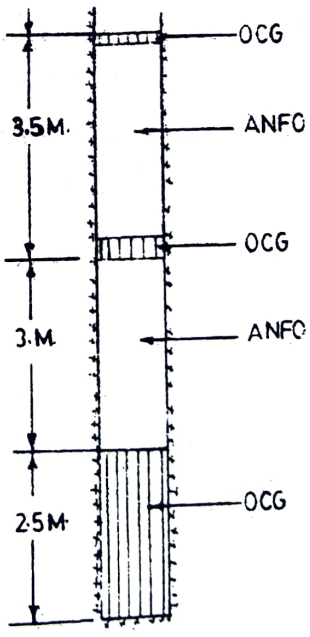


Fig. 43 Shows the distribution of various explosives in the blasthole.

from its operating characteristics, i.e., compressed air working pressure, blows per minute, stroke length and diameter of piston. Work output can be correlated with strength index of the rock in order to obtain penetration. For removal of cuttings the air pressure is to be increased which will also give more number of blows per minute. A bad blast can occur due to spacing and burden being improper, inadequate depth, inadequate charging, improper loading, improper connection of delay detonators, improper selection of explosives, etc. Difficulties of blasting may occur due to loss of drill meterage for leaving the holes for too long a period in comparatively loose or soft ground, collapse of holes due to movement of heavy mining machinery nearby or due to blasting of adjacent hole and also due to back break.

Question 121 : What environmental related damages may be expected due to blasting in the surface mines?

Answer :

Blast effects

The followings are the main environmental related damages which may be caused due to blasting in the surface mines :

- bench slope instability
- damage of vegetation by flyrock, dust, blasting-fumes, etc.
- instability to residential and other structures
- overbreak, back-break, etc. of the excavated area
- migration of wild-life,
- damage of materials due to flyrock
- effect on health due to noise, vibration, blasting-fumes, etc.
- alteration in ground water flow patterns due to change in micro cracks in the ground
- accidents to human and animals.

Question 121 A : How is noise produced during blasting in the open cast mines? What are it's remedial measures?

Answer :

Noise Problems

Blast oriented noise is produced mainly due to air-blast, poor blast design, initiation of detonating cord, firing of detonating fuse, poor

stemming of blast-holes, wind assistance blasting in hilly terrains, etc. Noise level greater than 140 dBL may cause damage to glass of windows and doors. Although noise due to blasting last only for a fraction of seconds but at hilly terains it lasts for more than 3 seconds due to echo effects.

Remedial measures : Cover detonating cord with 6'' thick layer of soil, use longer stemming column, use NONEL initiation system, make optimum blast design, avoid collar priming, use proper delay sequencing, don't conduct blasting operation during heavy wind and storm, etc.

Question 122 : What is the latest technique of assessing the blast performance?

Answer : Now-a-days high speed photographic techniques are widely used for assessing the blast performance like movement of fragmented muckpile from any particular location in the face. It can also nice]y be used to evaluate the sequence of initiation, velocity of the flyrock and ejected stemming materials. It also provides locating the misfired shot and gives a clear cut picture of colours of the blasting fumes with the help of a very high quality photograph taken by a high speed video camera with the help of high speed films.

Question 123 : How can blast performance be evaluated? Discuss.

Answer :

Some of the measures to evalute the blast performence :

1) Powder factor $= \dfrac{\text{Explosive consumption (Kg)}}{\text{Volume of broken rock } (m^3)}$

2) Percentage pull $= \dfrac{\text{Amount of pull (m)}}{\text{Drilled depth (m)}} \times 100$

3) Specific drilling $= \dfrac{\text{Meterage of drilled hole}}{\text{Volume of broken rock } (m^3)}$

4) Profile smoothness $= \dfrac{\text{Amount of over break/underbreak}}{\text{Amount of total excavation}} \times 100$

5) Bangs per Buck $= \dfrac{\text{Relative weight strength (RWS)}}{\text{Cost per Kilograms}}$

Where RWS $= \dfrac{\text{Absolute strength value of explosive}}{\text{Absolute strength value of ANFO}} \times 100$

The absolute strength value is a measure of the energy released (in Megajoules), from 1 Kg of explosives at a perfect and complete chemical reaction of the explosives constituents and that the explosive gases perform the useful work till they equalize to site specific normal temperature and pressure. Like that Relative Bulk Strength (RBS) is expressed as

$$RBS = \frac{\text{RWS of explosives x Sp gravity of explosives}}{\text{RWS of ANFO x Sp. gravity of ANFO}} \times 100$$

Sp. gravity of ANFO is taken as 0.8gm/c.c.

6) By assessing the loading machine performance

7) By larger boulder count.

According to the Kuz-Ram model, the mean fragment size can be calculated by the following equation

$$X = A \times (V/Q)^{0.8} \times Q^{0.167} \times (E/115)^{-0.633}$$

Where X = mean fragment size, cm

V = volume of blasted rock, m^3

Q = mass of explosive charge per hole, kg

E = relative weight strength of explosive (ANFO=100)

A = a constant based on rock factor (depends upon rock density, strength and jointing).

An estimate of the fragment size distribution is given by the Rosin-Rammler equation which is as follows :

$$R(x) = 1 - \exp\left\{-\left(\frac{X}{X_c}\right)^n\right\}$$

Where

R(x) = proportion of the material passing through the screen size x.

X = screen size, cm

Xc = characteristics size, cm

n = index of uniformity

$$n = \left\{2.2 - 14 \times \left(\frac{B}{d}\right)\right\} \times \left(1 - \frac{W}{B}\right) \times \left\{1 + \frac{(R-1)}{2}\right\}^{0.5} \times \left(\frac{L}{H}\right)$$

Where

d = charge diameter, mm

B = burden, m

W = standard deviation of drilling accuracy, m

$R = \frac{\text{Spacing}}{\text{Burden}}$

H = Bench height, m

L = Charge length, m

Value of n varies from 0.8 - 2.0

Priest and Hudson offer the following relationships on the probability of an intact length of rock being less than a specified size :

$F(x) = 1 - \exp(-aX)$

Where

a = mean fracture density,

X = distance between cracks, m.

8) By assessing the explosive consumption in secondary blasting.

9) By assessing the intensity of ground vibration.

The following ground vibration equation may be valid with a moderate accuracy :

$$V = K\left(\frac{D}{W^a}\right)^{-b}$$

where V = peak particle velocity, mm/s

K = site constant

D = distances of measuring point from the blast site, m

W = maximum charge per delay, Kg.

a = exponent

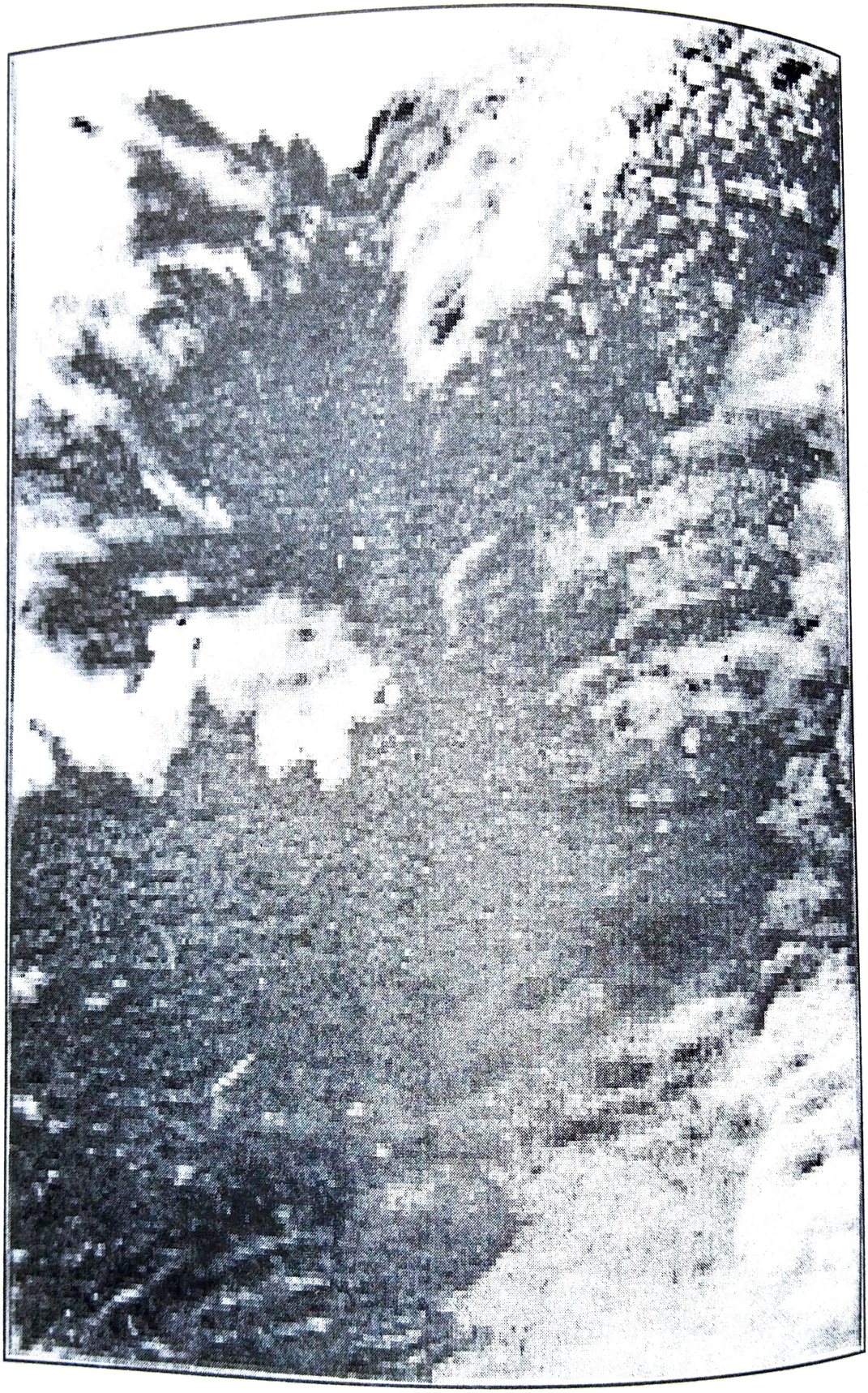
b = slope of best fit of V vs $\left(\frac{D}{W^a}\right)$ plot on a log-log scale

The above equation is further modified as follows :

Bis equation : $V = K\left(\frac{D^{2/3}}{W}\right)^{-b}$

Ambresay's - Hendron equation : $V = K\left(\frac{D}{3W}\right)^{-b}$

10) By assessing the fragmentation sizes.



By Courtesy : IDL Chemicals Ltd.

Question 124 : To ascertain the efficiency of blasting what are the various checks to be made for an efficient rock fragmentation?

Answer :

Various checks to be made for an efficient rock fragmentation

After blasting is over the efficiency of the blasting is ascertained with the following examinations and drilling and blasting pattern along with the other blasting parameters are changed accordingly. Questionnaire which are to be prepared for this purpose are as following :

- Whether blasting was uniform in heterogeneous rock formation throughout the length of hole after initiation of the explosive? If not why?
- Whether any deterioration at the bench, different bench slope and zig-zag curves are formed all along the line of bench edge? What are the reasons behind them and how much deviation was resulted from the original design?
- Does there any major back break formed in the block by the preceeding blasting? What are the reasons?
- Whether the drilling and blasting map given for drilling holes was accurate? What were the deviation of location, depth and direction of drilled holes?
- Does the inclination of holes matched the designed inclination? If not why?
- Was the blasting design method perfect? If not why?
- What is the size of the fragmentation? Is it in accordance with the requirement? If not why?
- What is the amount of yielding of oversized lumps? Was the proper energy given to blast the harder rock formation? Can the deck charging solve the problem of yielding of oversized lumps?
- What powder factor is actually obtained after blasting? What is the deviation from the designed estimation?
- What is the width of the muck pile, the spreading area of the individual lumps, etc.?
- What is the explosive consumption and cost of drilling and blasting?
- Does controlled blasting is required? If so then mentioin what

would be the firing pattern, the delay time, the number of rows, type of explosive, etc.?

- What are the major problems faced before the blasting operation? What are the reasons?
- What are the major problems arisen after blasting? What are the reasons?

During evaluating the performance of any blast the percentage of oversize in the fragmented muckpile, muckpile profile and displacement, amount of back break and overbreak, formation of toe, condition of remaining rock, vibration, flyrock, airblast, damage to slope walls, etc. should be examined in detail. The desired result will depend on the uses of blasting materials and accessories, excavation equipment, crusher sizes, etc. Performance of explosive is measured through velocity of detonation. Maximum fragmentation and muckpile displacement requires maximum release of energy from the explosive during detonation. Probes are used to measure the VOD of explosives in the blastholes. Inadequate priming, variation in composition of mixing, interference with water, etc. cause poor performance of an explosive. Shock energy caused by an explosive which causes vibration, is measured by the shock gauges. Size of the fragmentation can be judged by measuring the degree of mucking by the excavators.

The fragmented sizes directly indicates the basic objective of the blasting since it is the prime responsible for the efficacy of the rapid excavation and removal. Fragmentation size distribution may be assessed by the following method/models

(a) Automatic photography
(b) Video immage scanning
(c) Kuz-Ram model
(d) Sve DeFo model
(e) Composite model
(f) Analytical and semi-analytical techniques.

If fragmentation is fine or good, mucking and digging will be easy and the ore will flow smoothly from the draw point whereas if the fragmentation is coarse and rough mucking and digging by the excavators will be difficult and large or oversized rocks/ores have to be put a side for secondary breakage and also the draw point hangs up occasionally. The coarse and rough materials also be judged by heavy

steel plate wear, delays in crushing and will consume high power to crush the oversized materials. Recently photographs are used in the blasting muckpile to examine the degree of fragmentation by using a scaling object such as a ball of known diameter and these are compared with the standard photographs to find one which matches both in texture and size. The known scalling of both prints can then are used to evaluate the fragmentation in the blast. Analysis of fragmentation involves taking camera pictures of muckpiles, digesting the images and computer analysis for size distribution.

Recently high speed motion picture photography is used to study, analyse, evaluate and to aid in blast designs. Vibration monitoring by using computer, is used to evaluate the effectiveness of delay intervals in separating the individual vibration from blastholes and to determine the precise causes of high vibration which may be responsible for structural damage or environmental disturbance.

Question 125 : What are the main measures of formation of large boulders in the open cast mines and what are their remedial measures? Give an example.

Answer :

Formation of Large Boulders in the Opencast Mines and Their Remedial Measures.

The following are the main reasons of formmation of boulders and their remedial measures :

- Back break : Boulders those form behind the last row of blastholes are mainly due to back break. In case of back break the blasts on the bench pushes upward causing the top of the shot to lift up and break back sometimes as much length as a burden distance behind the last rows of blastholes. The upliftment results the formation of big boulders which commonly slide into the muckpile.

Solution :

(i) Use smaller diameter blastholes with reduced spacing and burden.

(ii) Reduce burden or do the optimum blast design based on the site specific condition, adopt control blasting with pre-splitting system, administrate optimum charge into the blasthole, etc.

(iii) Enhance the height of the bench.

- Large stemming zone : Large stemming zone containing no explosives, particularly in case of larger diameter blasthole, is responsible for the production of heavy amount of boulders.

Solution :

(i) Use smaller diameter blasthole with smaller burden, spacing and stemming length. It will facilitate use of higher explosive energy to break the rock causing less boulder formation but at the cost of higher drilling and blasting cost per tonne of production of mineral.

(ii) Use satellite charges in the stemming zone of the blast-hole and detonate it at the same time with the main charge with the help of common detonating cord. The satellite charge will be fired before moving any rock and the collar rock will be broken in proper place on top of the spot rather than they are blasted on top of the muckpile later on.

(iii) Use smaller diameter charge near the collar of the blast-hole and higher charge in the main explosive column. Do stemming surrounding the smaller diameter charge.

(iv)Drill additional blastholes at a lower depth in between the production blastholes, put charge inside of them, stem, and thereafter fire along with the production holes containing the main charge and/or satellite charge.

- Adverse geological condition : Because of presence of weathered zone, geological disturbances, presence of inherent cracks, presence of slip planes, etc. in the rock mass the latter fails in a predetermined manner after balsting causing a heavy amount of boulder formation.

  Solution :

  Do the Secondary blasting of big boulders over the muckpile.

- During blasting : It is due to poor blast design. The spacing and burden must be optimum. If spacings are very large boulder will definitely form. But if the spacings are very close then instead of getting in uniform energy distribution working outward from the blasthole a pre-shearing (similar to pre-splitting) will occur, energy will be lost and boulders will be formed profoundly in the burden in front of the blasthole. The

spacing, burden w.r.t. diameter of the blastholes must be the optimum for utilizing the explosive energy in most effective manner which will cause less boulder formation, better fragmentation and will also reduce the blasting cost.

- Wrong delay sequencing between the rows of blastholes : If delay between the rows are very short then the front row after firing cannot move horizontally outwards in time and as a result the back rows cause vartical cratering and uplift of rockmass in the form of boulders.

  Solution : Increase the delay timings between the rows of blast holes in the following manner :

In case of short height benches (bench height is less than 2 times the burden width) : 17 ms to 28 ms per metre of burden width

In case of high benches (bench height is more than the 2 times the burden width) : 12 ms to 15 ms per metre of burden width.

To reduce the size of the boulders the base charge may be loaded upto 40% of the total depth of the hole. But in this case loading of column charge is not needed.

Table-28 depicts the blasting practices in Joda East Iron ore Mines.

Table 28

Blasting Practices in Joda Iron Ore Mine

Parameters	**Rock Type**		
	Hard Ore	**Midium Hard**	**Soft Ore**
Average burden (m)	3.75	4.25	4.25
Average spacing (m)	4.25	4.25	4.75
Average bench height (m)	10.5	10.25	10.25
Diameter of hole (mm)	150	150	150
Base charge in the blasthole (Powergel-C) (m)	3.5	2.75	2.15
(Column charge used ANFO. Detonating cord used for blasting)			

Question 126 : What are the factors which govern the quality of raw material, degree of rock fragmentation and formation of lump size?

Answer : The factors which are responsible for quality of raw material, degree of rock fragmentation and lump size are as following :

- Blast hole parameters like diameter, depth, direction. subgrade drilling, angle of inclination with the horizontal, etc.,
- The pattern of blast holes,
- The explosive parameters and quantity of explosives,
- Method of blasting and volume of yield of rock after blasting,
- Efficient organisational network for blasting, etc.

Question 126 (A) : What are the main reasons of poor blast? What are their remedial measures?

Answer :

The main reasons of poor blast and their remedial measures are as follows—

- poor blast design : Design optimum blast as per the site specific conditions.
- wrong delay sequencing : Select correct delay sequencing taking all due consideration of very good blast but with nominal air blast, noise, ground vibration and boulder formation.
- adverse gelogical conditions : It is an uncontrollable factor. But the blast design must be in the tune against all adversity.
- wrong selction of explosives : Poor quality, low energy and low VOD are also responsible for bad blast :
  Select proper type of explosives. Never use old and poor quality explosives.
- improper selection of the right type initiation system : Select the right type initiation system which includes detonators, boosters, fuses, cords, electric wires, etc.
- Due to drill hole deviation : It is mainly caused by the poor surveying, positioning and supervision. The inhole deviation can be reduced by careful drilling, alignment, right choice of drilling equipment, accurate collaring, avoiding the zone of irregular cracks and weakness planes, etc.
- due to improper charging of the blast-hole : Adopt good

charging procedure, use explosive loading machine for charging the blast-hole accurately, etc.

Question 127 : What are the main factors which control the efficiency and cost of drilling and blasting? How the blasting cost can be reduced in an opencast mine?

Answer :

Efficiency of Drilling and Blasting

The drilling and blasting efficiency depends on mainly on the parameters of drilling, pattern of drill holes, type of explosives and explosive parameters, drillability and explosibility of the rock, watery condition, etc.

Cost of Drilling and Blasting

The drilling and blasting costs are dependent on the powder factor, the drilling cost per metre length of hole, the cost of explosive per kilogram yield of rock per metre of drill hole, the consumption of explosive in kg/m^3, cost of blasting means, charging, stemming and firing, quality of blast, optimal design of drilling and blasting pattern, etc.

Blasting performance is mainly governed by the results of fragmentation, powder factor obtained and the cost of blasting. The blasting cost can be reduced by the following :

- by consuming low cost bulk explosive system in opencast mining in place of costly cartridge explosives. It will also reduce drastically the storage and inventory cost, manpower cost, heavy capital expenditure on magazine and explosive van, etc.
- Locating explosive magazine near to the mine, will lower the transport cost of the explosives,
- powder factor will increase if the explosive with less loading density is used,
- drilling cost will reduce if higher spacing and burden is designed. It is possible if only explosive with high loading density is used. Since cost of primer is more hence decrease in ratio of primer to non-primer explosive will reduce the explosive cost.
- If higher energy explosives are used it will increase the powder factor and hence reduce the blasting cost,

- good fragmentation will cause good effect on loading, hauling, crushing etc. Hence oveall mining cost will reduce. However, if the fragmentation is poor then it will adversely effect loading, hauling and crushing system and heavy amount of secondary blasting will be necessary. Bad fragmentation will also result heavy wear in the buckets of shovel and dragline, rope of the dragline, damage of tyre and bodies of thc truck, etc.
- If trucks are used for transporting and loading bulk explosive system then it will reduce the loading cost.

  By using air decking technique, using low density explosives, the blast hole pressure in the moderately stronger rock will decrease and explosive cost will reduce;
- in watery mines if boreholes are first dewatered properly before charging the explosive, the blasting efficiency will be improved and thc explosive cost will decrease,
- large diameter blast hole will enhance higher quantity of fragmentation of rocks, reduce operating cost and cost per tonne of mineral or rock produced,
- angle hole drilling improves powder factor, reduce formation of toe, back break and much amount of boulder formation. It will also reduce the overall consumption of explosives.
- the drilling operation in a particular patch in a bench should be completed and blasted as fast as possible otherwise there will be the problem of caving of holes. And once the holes are caved, there will be various other operational problems and ultimately the drilling and blasting cost will incrcase;
- optimization of blasting parameters, selection of explosives and accessories depending upon the geotechnical parameters of rock, explosive properties, drilling parameters, excavator size, etc. will reduce the overall blasting cost. However, the blasting cost also should consider the environmental impact, damage of benches and other operating cost which are indirectly influenced by it e.g., loading system, transporting system, crushing system, etc.

Question 128 : Give some realistic idea about the performance of blasting in cubic metre of rock produced/kg of explosive or production of

ore/coal in tonne/kg of explosives in both underground and opencast metal and coal mines.

Answer : If the standard type of blasting pattern is practised then with the application of optimum charge on an average the following realistic figure (Table-29) can be obtained.

Table 29

Shaft Sinking		
In coal measure rock (blasting with permitted explosive)	Cubic metre of rock/kg of explosive (Depending upon the hardness of rock)	0.55 to 1.3
In metalliferous mining deposit	- do -	0.35 to 0.7
Underground coal mines (Blasting with permitted explosives)		
Drifting in stone	Cubic Metre of rock/kg of explosive (Depending upon the hardness of rock)	0.7 to 1
Development in coal		
Blasting off the solid	Tonne/kg of explosive (depending upon hardness of coal)	2.4 to 3.0
Blasting with a cut by CCM	Tonne/kg of explosive (depending upon the hardness of coal)	5 to 7
Blasting with Auger drill	Tonne/kg of explosive (depending upon hardness of coal)	2.7 to 3
Extraction		
From bord and pillar working	Tonne/kg of explosive (depending upon the hardness of coal)	4 to 5
From longwall working (with solid blasting)	Tonne/kg of explosive (depending upon the hardness of coal)	4 to 5
From longwall working with a cut by CCM	- do -	6 to 8

Table 29 (Continued)

Opencast Coal Mines			
Overburden	With LOX and 60 % SG	Cubic Metre per kg of explosives (depending upon the hardness of rock)	4 to 6
	With Aluminised cartridge slurry and emulsion with NG explosives in the deep hole blasting	- do -	1.5 to 3.0
Coal	With LOX and 60% SG	Tonne/kg of explosives	7 to 8
	With OCG	Tonne/kg of explosives	8 to 10
Opencast Mining in Metalliferous Deposits			
Iron Ore	With slurry explosives and OCG booster	Tonne/kg of explosives	4 to 5
	With GN_1	Tonne/kg of explosives	4 to 6
	With LOX	Tonne/kg of explosives	5 to 7
	With OCG	- do -	6 to 8
Limestone	With OCG	Tonne/kg of explosives	6 to 9
	With slurry explosives and OCG booster	- do -	4 to 5
Copper ore	With OCG	Tonne/kg of explosives	6 to 8

Table 29 (Continued)

Underground Metalliferous Deposits			
Drifting	SG 80%	Tonne/kg of explosives (depending upon the type of rock)	0.8 to 1.5
Winze	SG 80%	Tonne/kg of explosives (depending upon the type of rock)	0.8 to 1.4
Raise	Hectorite or SG 80%	- do -	0.8 to 1.5
Stoping methods (with conventional drilling & blasting)	S.G. - 60%	- do -	3 to 4
Stoping methods with longhole drilling & blasting	ANFO with booster charge	- do -	2 to 3

Question 129 : Discuss methodologies of calculation of drilling and blasting cost.

Answer : The drilling and blasting efficieney depends upon the method of drilling, diameter of the drill hole, blast hole pattern, type of explosives, drillability, explosibility, geo-technical permaters of rock, etc. The drilling and blasting costs are dependant on the cost of drilling per metre of hole, the powder factor, the cost of explosives/kg, yield of ore/ mineral/rock per metre of drill hole, etc.

Suppose a mine is desired to produce 'A' tonne of ore from a single bench per day.

So production per week = 6A tonne or $\frac{6A}{d}$ m^3

Where d = density of the ore in tonne/m^3.

So for this block of ore($\frac{6A}{d}$ m^3) let the sum of the production cost in drilling during that week is R_1.

Therefore, drilling cost/m^3 of ore = $\frac{R_1}{\frac{6A}{d}} = \frac{dR_1}{6A} = C_1$ (say)

If the drilling cost of 1 m of hole = r_1

and a is the yield of ore/m of hole length

then the drilling cost/m^3 of ore = $\frac{r_1}{a} = C_1$

$$\therefore C_1 = \frac{dR_1}{6A} = \frac{r_1}{a}$$

The above cost of drilling is obtained from the addition of the following main costs :

(a) wage cost of the driller, helper, compressor operator, other labourer, supervisor, officer (if any), etc.

(b) Depreciation cost, maintenance cost, interest on working capital, taxes and insurance cost (if any) of the drill machine, drill bits and drill rods, compressor and other drilling accessories.

(c) Store cost e.g., fuel, lubrication, etc.

(d) Power cost, etc.

For calculation purpose the following norms may be followed :

Wage cost

Average earning per manshift (category wise) X Total number of workers (categorywise)

= Total wage cost of worker ... (i)

Average earning per manshift of supervisor X Total number of supervisors

= Total wage cost of supervisors ... (ii)

Average earning per manshift of officer X Total number (if any) of officer

= Total wage cost of officers ... (iii)

Total wage cost = (i) + (ii) + (iii) ... x_1

Depreciation, Interest on Working Capital, Taxes and Insurance Cost, etc.

Now let the cost (delivery price) of a single equipment like crawler drill machine is K. If the life of the drill machine is 10 years, and the actual working days in a year is 300 days, then the depreciation cost/day on straight line method is

$$\frac{K}{10x300} = \frac{K}{3000} = 0.000333K.....x_2$$

Let maintenance and rapair cost per day is 50% of the depreciation

$$\text{cost} = \frac{1}{2} x \frac{K}{10x300} = \frac{K}{6000} = 0.000166K.....x_3$$

Interest on working capital is 14% of average annual investent.

$$\text{Average annual investment} = \text{Delivery price} \times \frac{N+1}{2N}$$

Where N = 10 years.

$$= K \times \frac{(10+1)}{2 \times 10} = \frac{11}{20} K$$

$$\text{So interest per day} = \frac{0.14 \times \frac{11}{20} K}{300} = 0.000256K.....X_4$$

(If taxes and insurance is to be paid which may be 5% of average annual investment then taxes and insurance cost/day

$$= \frac{0.05 \times \frac{11}{20} K}{300} = 0.000916K.....X_5$$

Store and Other Material Cost.....x_6

Average consumption of fuel = 0.225 litres/h.p./hour. Oil and lubrication

$$\text{cost} = \frac{1}{3} \text{ x Fuel Cost}$$

Cost of bit, drill rods and other material is roughtly 18 to 20% of the equipment cost.

Depreciation cost of the drill bit, roughly varies from 35 to 40%

Power Cost.....x_7

Average power cost/day = Tatal KW of the machine x Hours worked per day x Unit cost.

So total drilling cost/day = $x_1+x_2+x_3+x_4+x_5+x_6+x_7 = X$ (say)

Blasting cost per cubic metre of rock

Let production cost of blasting during that week = R_2 and volume of rock blasted $\frac{6A}{d}$ m^3

(Here for the ease of calcution volume of drilled block and the volume of ore blasted is considered to be the equal.)

So blasting cost per cubic metre of ore

$$= \frac{R_2}{\frac{6A}{d}} = \frac{dR_2}{6A} = C_2 \text{ (say)}$$

If the average waighted consumption of explosive is e kg/m^3 and m is the avertage cost of explosive per kg and n is the average tatal cost of blasting accessories, charging, stemming, transport of explosive, wage of blaster helper and other labourer, firing etc.

Then, $C_2 = em + n$

Generally n = 15 to 20% of em.

So $\frac{d\,R_2}{6A} = e\,m + n = C_2$

Therefore, blasting cost/tonne of ore = $\frac{C_2}{d}$

So blasting cost/day of ore production = $\frac{C_2}{d}$ A..... Y(say)

Y also can be calculated separately by adding up explosive cost, blasting accessories cost for daily ore production; daily wage cost of blasting personal, daily equipment's (like explosive van, slurry pump truck, etc.) depreciation cost, daily interest on working capital of those equipment, daily taxes and insurance cost of those equipment, daily store and other material cost for blasting work, etc. Let after adding X and Y the total value become X + Y = B. With the value 'B' work and company overhead cost is to be added. The work overhead may include the cost of depreciation, interest on working capital, repair and maintenance, taxes and interest, etc. of the blasting engineer's and foreman's jeep and other vehicle (if any), cost of site office, miscellaneous job etc. The company overhead cost would include the wage and other cost of office personal, workshop personal, general supervisor, general officer or any interest cost. So with 'B' 15% job overhead and 19% for company overhead is to be added.

Therefore the total cost/day comes to 1.3 B.

Hence drilling and blasting cost per tonne of ore = $\frac{1.3B}{A}$ or cost per cubic metre of ore = $\frac{1.3B}{A}$ d

However, the drilling and blasting cost is variable depending upon the drillability and explosibility of rock.

With this analysis gathering of a generalised idea of a consistent, optional quality of drilling and blasting in rocks of each category at a minimum cost of mining is possible.

If overburden is to be also removed, depending upon the stripping ratio separate cost calculation for the overburden is to be made in the same manner.

Just for the purpose of calculation the following data may be used—

Let mine is to be produced	: 1 million tonne of ROM/year
Bench height of an iron ore mine	: 12 m
Diameter of the hole	: 150 mm
Inclination of the hole	: vertical
Burden	: 4 m
Spacing	: 4 m

Tonnage per hole : 4m x 4m x 12m x 3.4 = 652.8 tonne. Output per week from the mine = $\frac{1\times10^6\times6}{300}$ = 20000 tonne

Therefore number of holes required/week = $\frac{20000}{652.8}$ = 30.6 = 31

Total drilling per week = 31 x 13.5 = 418.5 m

(Considering depth of hole with 10-20% subgrade hole in a 12 m bench = 13.5)

Anticipated drilling with 12 hours of operation/day = 80 m. So one drill can drill 80 x 6 = 480 m/week.

Assuming 70% availability and 80% untilisation, number of drills required will be $1\times\frac{100}{70}\times\frac{100}{80}$ = 1.78 = 2 nos. (say)

Considering 1 spare drill machine the total number of drill machine required = 2 + 1 = 3 nos.

Other machinery

Slurry pump	: 1
Explosive van	: 1
Engineer's jeep	: 1

Assume the following manpower per day

Drilling Manpower	Blasting Manpower
Driller 2 x 3 = 6	Supervisor -1
Driller helper 2 x 3 = 6	Blaster - 1
Supervisor 1 x 3 = 3	Blasting helper -2
Maintenance 2 x 3 = 3	Explosive van driver - 1
	Pumptruck driver - 1

In General shift

Engineer : 1

Jeep Driver : 1

The drilling and blasting operation will be also partially backed by the staffs of the site office, workshop, main office, etc.

Approximate charged column : 10 m

Approximate stemming height : 3.5 m

Primer : Toe blast, charge/hole (1.5 m column height approx.) = 18.75 kg

Main charge : SMS (8.5 m column height approx.) - Charge/hole (approx.) = 174.25 kg

Total charge (approx.) : 193.00 kg

Tonnage of production/kg of charge = 3.38

Assuming Cost of primer charge (approx.) : Rs. 25/kg

Assuming Cost of SMS (approx.) Rs. 17 per kg (200 price)

Assuming approximate cost of one 150 mm dia drill machine including all accessories and spares Rs. 5000000.00 (1992 price)

Assuming approximate total cost of explosive van, pump truck and jeep (including all accessories and spares) Rs. 1500000.00

With the help of the above data and procedure it will be very easy to calculate the Drilling and Blasting cost/tonne of ore.

In the drilling and blasting cost in the surface mining system drilling cost accounts around 46% and blasting cost accounts around 54% of the total cost.

Explosive accounts around 3 to 8% of the total production cost of per tonne of mineral or coal.

Question 130 : Give some approximate idea regarding the cost of the explosives and other blasting accessories.

Answer : Approximate cost of the explosives are given below :

Explosives	**Approx. cost per kg. (year 1999) Rupee**
1. Permitted explosives (small dia) for underground use.	35-40
2. Non-permitted explosive (large dia) (for opencast use)	20-25
3. Small dia. special Gelatine	35-40
4. Bulk explosives	12.5-20
Detonators and Detonating fuse	**Approx. cost per piece, rupees**
1. Electric delay detonator 2.5 to 3 m lead wire for use in gassy mine	10.5
2. Copper electric detonators with 1.8 m lead wire	9.0
3. Allminium long or short delay detonator with 3m lead wire	10.0
4. Alluminium electric detonators with 1.8 m lead wire	9.0
5. Detonating relays	8.0
6. Detonating fuse	3.5-4.0 per metre
Exploder	**Approx. cost in Rs. per unit (year 1999)**
1. Multishot exploder, intrinsically safe	12000
2. 100 shot, cell life 300 shot, energy 620 MJ Hg cell, 5.6V, weight 0.8 kg.	4000
3. 25 shot dry type star battery operated for opencast mine	8000
4. 100 shot dynamo type for opencast mine	11000
5. 200 shot dynamo type for opencast mine	12000

Question 131 : Write brief notes about the controlled blasting techniques. What is presplitting technique? How is it done? Discuss elaborately about the air decking technique. Write step by step about the general procedures in :

a) air deck mothod of presplitting

b) multi deck inhole delay blasting technique.

Answer :

Controlled Blasting Techniques

Increasing use of techniques to control damage to the remaining rock is being practised in surface mines. Smooth blasting introduced in Sweden uses thin pipe charges like Gurit and Babit, consisting of power explosives. In smooth blasting light decoupled explosives are used with small burden and fired as the last delay. In pre-splitting holes in the last row is detonated in the beginning, thus creating a cut between holes. In cushion blasting introduced in Canada involves a simple row of holes along the neat excavation line. The holes are loaded with light well distributed charges completely stemmed and fired after the main excavation is removed. For many operations ANFO has been used with light polysterine granules to reduce effect of explosives.

Presplitting

Presplitting creates an artificial discontinuity along the periphery of the designed and expected limit of the main blast fragmentation to isolate the blasted zone from the remaining rockmass. It helps to prevent overbreak and creates smooth and stable highwall, and also help reflection of shock waves from the subsequent main blast. This can be done in two ways :

a) by in hole delay arrangement and

b) by air deck method of presplitting holes.

Air Decking Technique

When an air-gap is introduced in the midst of an explosives columns, explosion products from two portions of the charge generate shock waves (Fig. 44, 45, 46) which move in opposite directions. In the

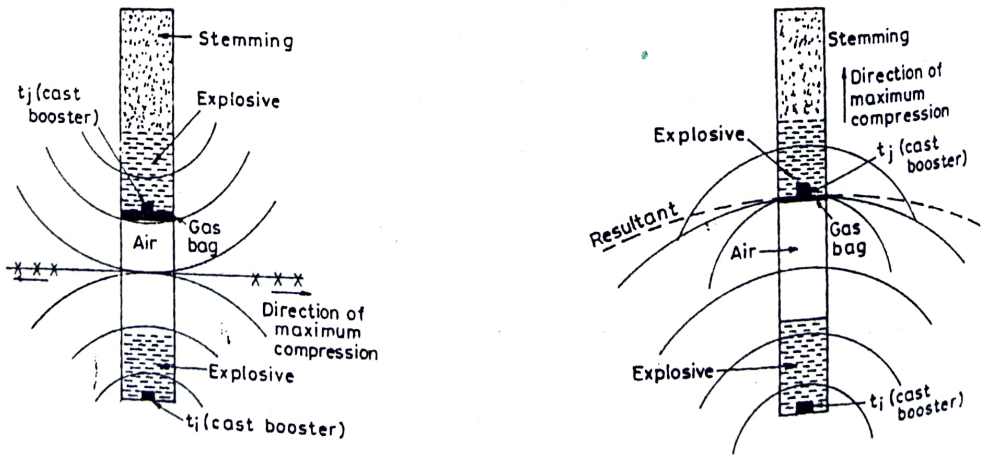


Fig. 44-45 Collision of shock waves in the Air-gap

middle of the air-gap there will be a collision of the shock waves. As a result of this collision, a new source of high pressure is formed in the centre of the air-gap. After reflection, the shock waves will change direction and start moving towards the bottom of the hole and the stemming. Reflected from these hard obstacles the shock waves will again collide in the centre of the air-gap and the process will be repeated. This results in better utilisation of explosives energy and uniform fragmentation throughout the length of the borehole.

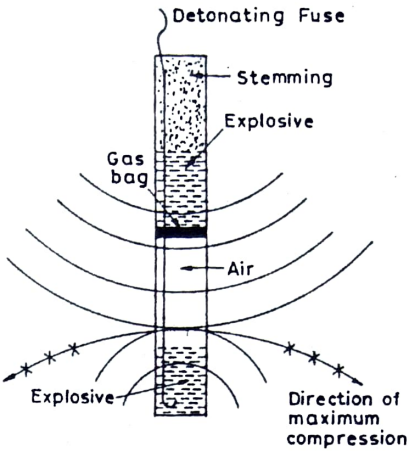


Fig. 46 Collision of shock waves in the Air-gap

Air spaces and air gaps in an explosive column are known to allow greater distribution and cushioning of explosive from the rock mass. This energy distribution is enhanced if the explosive gases can be confined in the boreholes for as long as possible by plugging and stemming above the air space. Power plug is an inflatable plug which can seal a borehole at any point to separate the stemming from an air gap and explosive toe load.

The air decking technique using inflatable power plug can be used for ensuring pit wall stability and selective mining of coal seams by predrilling under seam waste wedges. Air decking techniques require that an air space be created in the borehole above the explosive charge. This air space allows gases from the explosion to expand and spread throughout the length of the hole so that damage to adjacent rock is minimized (Fig. 47). It will also cause less localized shattering of rockmass. But with the complete stemming of the blastholes causes prevention of gas venting into the weak strata and tearing the rock apart. This shall be the better technique in case of badly fractured ground. The crack formed by the presplitting should be at least 5 mm in width.

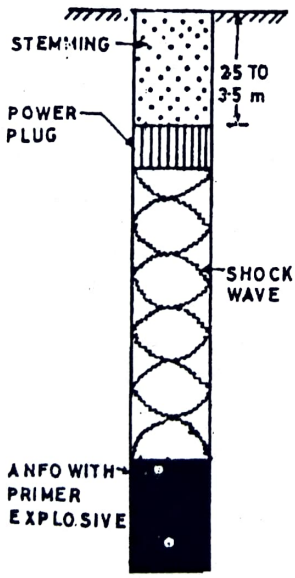


Fig. 47 Airdeck method of controlled blasting in pre-splitting

General procedure in air deck method of presplitting is as following :

- First estimate the burden and spacing by measuring the tensile strength of the in-situ rockmass.
- Drill holes of 200 to 250 mm diameter up to a depth varying from 10 m to 30 m as per the design requirement. Spacing in between the holes may be 5 to 12 times thc diameter of the holes. The distance of the production hole is generally 3.5 times the diameter of the hole.
- Pour ANFO with primer charge upto a depth of 1/10th of the depth of hole. Explosive charge generally 0.9 to 1.3 kg/m^2 of split.
- Cover the holes with the power plug near to the mouth of the holes.
- Stem the holes say upto a height of 3 m with the stemming material as mentioned earlier. Length of the stemming column should be at least 3.5 to 4.0 times the diameter of the hole.
- Blast nearly 50 m length of the bench at a time.

The presplitting will eliminate the back breaks on the high wall side, there will be increase in permissible charge 300 to more than 400 times and also reduction in ground vibration level more than 70%. Presplitting also stabilize the highwall, ensure high percentage of coal or mineral recovery, ensure safety to both man and machine working in the mine and also extends safety to inhabitants in the close proximity of surface coal mines.

Multideck Inhole Delay Technique

In multideck inhole delay blasting the arrangement is done in the following manner :

- First estimate the burden and spacing of holes by measuring the tensile strength of the in-situ rockmass.
- Drill holes of required diameter upto the design depth.
- Insert primer explosive first into the hole along with primer.
- Detonating Realy Cords (DRC) may be used for firing if only one delay is to be provided for blasting a hole. DRC may be available at 15, 20, 25, 42, 50, 100, 150 ms delay periods.
- After pouring first primer explosive, stem the holes with stemming material.

- Insert primer explosive 2nd time over the stemming material along with primer.
- Stem the remaining portion of the hole with stemming material.
- Continue to charge the holes by alternate stemming and primer explosives as per the design.
- Blast the holes.

Multideck inhole delay blasting will reduce the vibration level and maximise charge per delay (Fig. 48).

It is worth mentioning that pre-splitting shotholes may also be blasted with the decoupled explosive cartridges which are attached with the detonating cord with specially designed connector. The charges in a shothole should be blasted simultaneously with the help of either detonating cord or electric detonators, whereas number of presplit shotholes are to be blasted with the use of short delay detonators. To reduce the overbreak the charges shall be loaded as close to the collar (after duly considering the fly-rock problem) and in case of unconsolidated rock formation charging may be done upto the collar. The open collar permits the free escape of the gases produced and imparts lesser shock loads to the surrounding rockmass and also the less amount of overbreak. In between the two charged holes a relief hole may be drilled to promote shear along the desired plane. It will also help to reduce the backbreak.

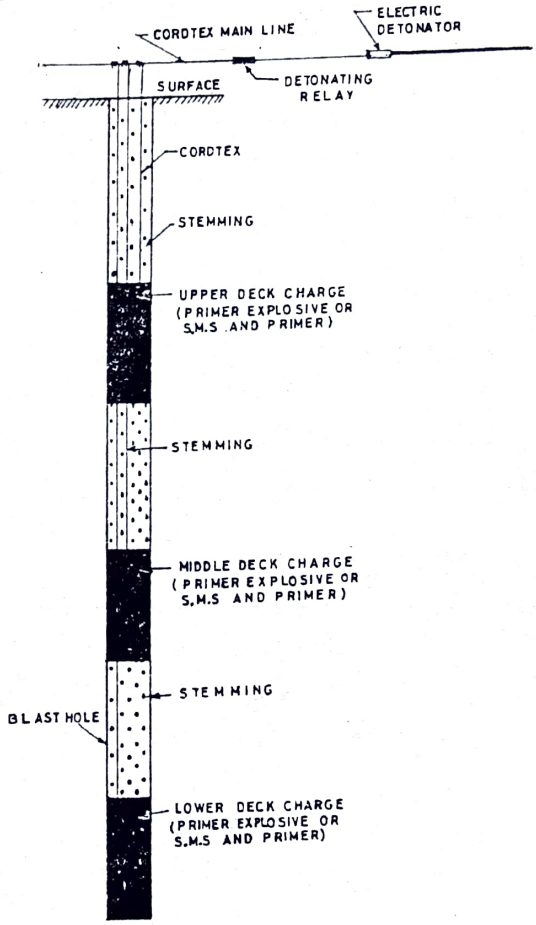


Fig. 48 Multi-Deck in hole delay method
(Note : The main line can be initiated either by an electric detonator or by safety fuse and plain detonator)

Question 132 : Discuss the blasting practices in the Kudremukh iron ore mines using the air decking techniques.

Answer :

Blasting Practices in Kudremukh Iron Ore Mines

The followings are the main theme of blasting practices in the kudremukh iron ore mines (Table - 30).

Table 30

	Soft Ore	Medium Hard Ore	Hard Ore
Compressive Strength (MPa)	190-200	201-215	216-225
Spacing (m)	11	10	9
Burden (m)	11	10	9
Length of blasthole (m)	17	17	17
Explosives used :			
Bottom charge	SMS. 674 or 654 series of IBP		
Rate of pumping	182 Kg/minute		
Column charge	Emulsion explosives of KEL		
	2.5% to 3% Al		
Rate of pumping	120 Kg/minute		
Booster used	Cast Booster (0.1%-0.2% of the explosive charge)		
Borehole sleep time	2 weeks		

SMS = Site mixed slurry explosive
IBP = Indo Burma Petroleum
KEL = Keltech Energies Ltd.

The table - 31 shows the main comparision among the blasting methods practised in the said mine.

Table 31

Parameters	**Conventional blasting method**	**'Opti-Blast' with cane Baskets (Air-Decking Technique) (Fig. 49)**	**'Split-charge' with Gas-bags (Air decking Technique) (Fig. 50)**
(1)	**(2)**	**(3)**	**(4)**
Length of hole (m)	17	17	17

Table - 31 (continued)

1	2	3	4
Bottom Charge			
Length (m)	5.7	3	3.7
and Weight of charg (Kg)	485	219	315
Cast booster Weight (Kg)	0.5	0.5	0.5
Column Charge			
Length (m)	3.7	5	3.7
Weight of charge (Kg)	315	366	315
Cast booster weight (Kg)	0.5	0.5	0.5
Cast booster requirement	——0.1 to 0.2% of the explosive charge——		
Stemming height (m)	6	6	6
Stemming materials	——Drill cuttings——		
Air decking height (m) and methodology	-	1.4 (varies from 1-3 m) In the top of the column between the Column charge and stemming by suspending a cane basket from the collar of the hole.	2.0 (varies from 2-3.5 m) Air deck is created in the middle of the explosive column and gas bags are used in place of cane baskets
Back filling height (m)	1.6	1.6	1.6
Powder factor (gm/ton)	230-288	202	195
Basic Comparision			
Effectiveness of maintenance of air-gap	-	Not very much effective	Very effective
Fragmentation	Not satisfactory	Satisfactory	Very good
Flyrock	Severe	Less	Negligible
Ground vibration	High	Within the statutory limit	Very low and less than optiblast
Presence of water in the blasthole	Suitable	Unsuitable	Suitable
Explosive energy utilization	Poor	Good	Very good
Cost	Expensive	Cheaper than conventional method	Most economical

Note : In air-decking technique the gas-bag which is being used by the kudremukh iron ore company limited, is made of multiple layers of Nylon Co-polymers and contains an Aerosol can with a plunger. The capacity of the can is 200 c.c. and it contains 100 gm of Propane gas. One tag is provided on the top of the bag for lowering it down into the blasthole, using a string. As soon as the plunger is pressed, Propane gas is released from the can and it inflats the bag within 45 second, and becomes tight fitted with the blasthole at the pre-determined depth. Then the tag is pulled out. Air decking technique will facillitate the better utilization of explosive energy by the collision of explosion-gas-waves, gives rise to secondary breakage at the mid-point of air gap and continuity in transmission of explosive energy throughout the air-column.

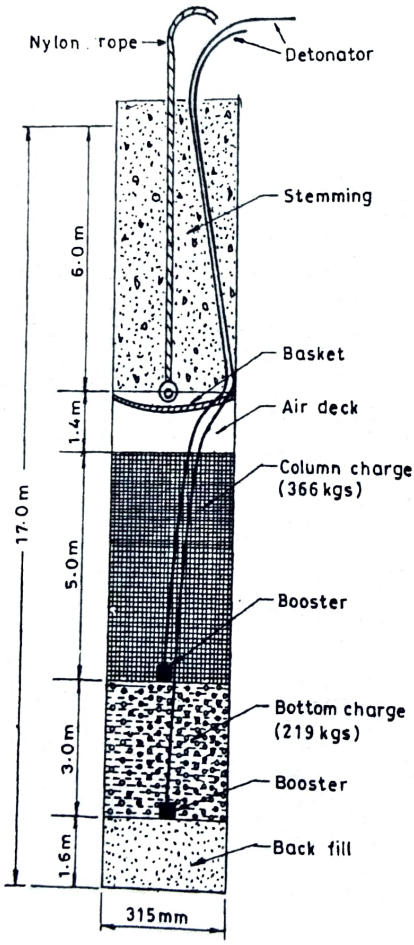


Fig. 49 OPTI-Blast Technique

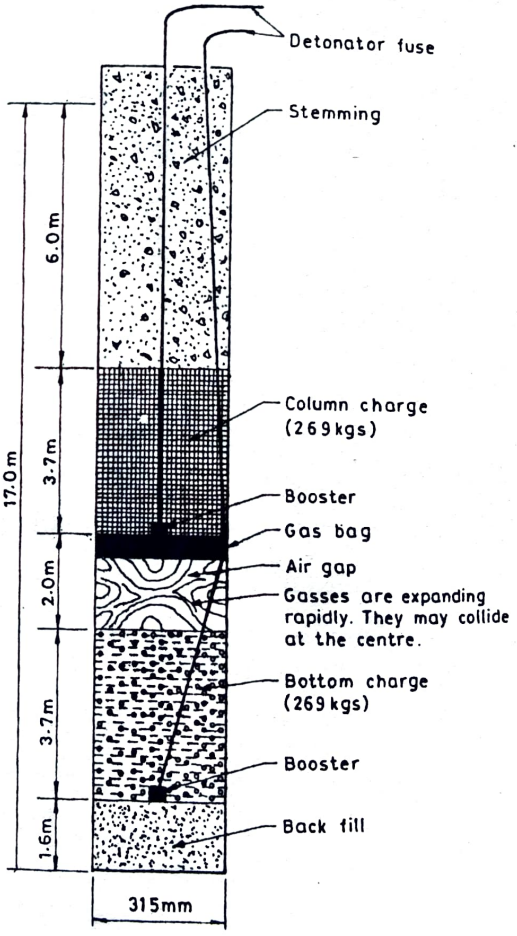


Fig. 50 SPLIT-Charge Technique

Kudremukh iron ore Co. Ltd. used three types of firing sequence :

i) 'Parallel' cut : In case of bench length is very high but bench width is limited for only 3 rows of

blastholes then this pattern of cut (Fig. 51) is being followed.

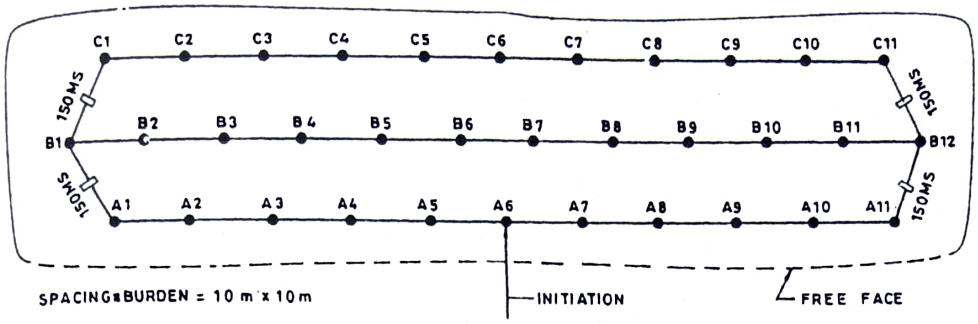


Fig. 51 Parallel cut Pattern

ii) 'V' cut : This type of cut is used if the bench length to width ratio is 3:2. In this cut try to avoid acute angle to prevent misfires/ cut-offs. Number of rows should be limited to 3 or 7 and the total firing time is limited to 3000 ms (Fig. 52).

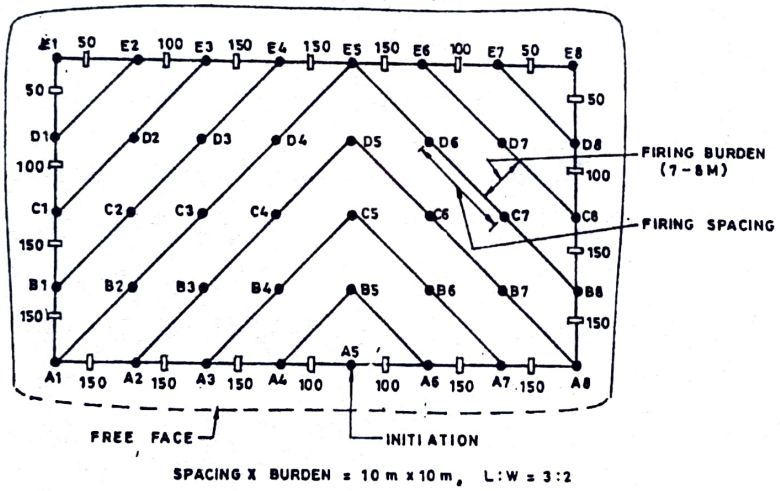


Fig. 52 'V' Cut Pattern

iii) Trapezoidal-cut : This is better than the parallel cut and V-cut since the trunk line does not form acute angle, offers good fragmentation, reduce percentage of formation of boulders less than 0.5%, does not from toe, achieve better throw, reduce secondary blasting and since firing burden is less than the actual burden it offers better fragmentation.

In this system hole to hole delays are maintained 50 ms and row to row delays are maintained 150 ms. For better fragmentation 3-5 ms delay is required for every metre of burden (fig. 53). During proper blast design it must be ensured that before a particular hole gets detonated, the holes in front and side are moved by approximately 5%-20% of the burden and 2%-10% of the spacing distance.

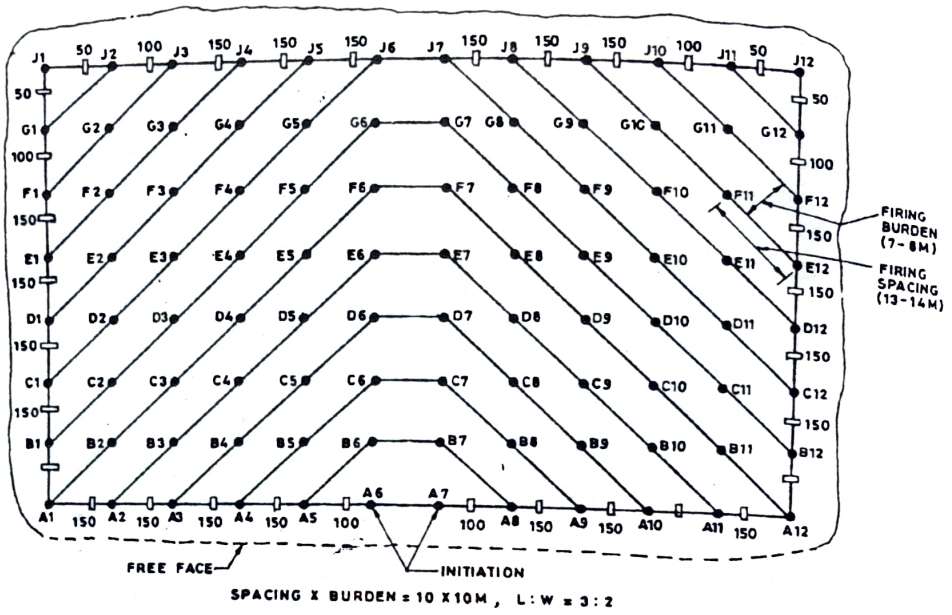


Fig. 53 Trapezoidal Cut Pattern

Question 133 : Briefly give an example of air decking technique using gas-bags in an open cast coal mine.

Answer :

Result of Air Decking (Using Gas-bags) in Open cast Coal Mines

Hole diameter (mm)	:	311
Strata	:	Medium hard sandstone
Burden (m)	:	8
Spacing (m)	:	11
Bench height (m)	:	17
Explosive used	:	Bulk loaded slurry explosive
Charge per hole with air decking (kg)	:	670-700
Result	:	Excellent and uniform fragmentation, no back-break, negligible throw, less fine production.

Question 134 : Describe a Gas-bag used for air-decking technique.

Answer :

Air decking with Gas bags

A gasbag is a self in-flating bag made of high strength multi-layer co-extruded nylon co-polymer films with an Aerosol can inside containing the inflating gas. The aerosol can has a trigger at the top of it. When the trigger is activated it releases the content inside the aerosol can into the gasbag. After activating the aerosol can on the surface, the gasbag is lowered down at the desired position of the hole by a string. The gas releasing from the aerosol can inflates the gasbag and ultimately firmly be locked inside the borehols. The space below the gasbag is the air deck. After putting the gasbag in the desired position the string is pulled causing the tag break. The gasbag is ready to carry the high load of explosive column. The inflatting gas in the aerosol can is moderately inflammable and hence the complete unit shall be handled very carefully. The unit shall not be kept for a longer time in the sunlight, must be kept away from the naked flame, and the unit shall not be punctured or burnt. If the unit is starting inflatting on the surface, then the bag shall be punctured/cut in a well ventilated area away from any naked flames. The gasbags may be used during presplitting, controlled blasting, sealing of leakages, during using of ANFO in wet holes, for protecting bulk explosives from contamination by heavy drill cuttings in iron ore strata, for reduction of fines, for improving performance of bulk loaded slurries, etc.

Gas bags are available in a range of sizes to suit blast hole diameters from 75 mm to 400 mm. The high strength multi-layer coextruded nylon polymer construction of the bags allows the inflated units to carry high loads of column of explosives or dense stemming materials for a longer periods of time. The construction is designed to handle most situations which may occur in drilling and blasting operations and the air-decked holes can withstand sleeping of several days. (Fig. 54)

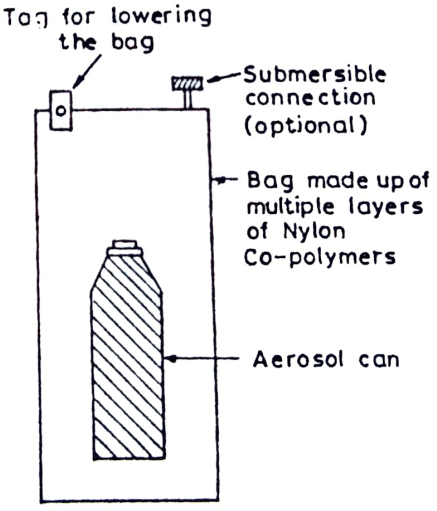


Fig. 54 The Bostech Gasbag

The bag has approximately a 20 sec. delay before inflation begins. Full inflation can take up to 2 minutes for large hole units.

Charging Gas Bags

(i) Calculate the quantity of explosive and lenght of column for bottom deck, top deck and length of air-deck.

(ii) Determine suspension point for gas bag and string length.

(iii) Charge bottom deck of explosive and prime it as usual.

(iv) Activate gas bag and suspend it into the borehole at the required level by string.

(v) After inflation is completed, pull string. Tag breaks and gas bag is locked in position.

(vi) Fill top charge of explosive.

(vii) Stem at top as usual.

(viii) Hook up with necessary delay patterns and initiate as usual.

Gas bags can be efficiently used in the following applications :

- Controlled blasting,
- Pre-splitting
- For improvement of performance of bulk loaded slurry explosives in case of very large deep holes. Use of gas bages in between the long explosive column (i.e. in a very high hydrostatic head) will reduce dead press desensitisation of slurries.
- Sealing : The gas bag can efficiently be used over the open borehole, and cracks, so that the slurry explosives do not leaks through them. Hence a large quantity of explosives can be saved in this manner.
- In wet boreholes : The gas bag can be placed atop water into a blasthole and ANFO can efficiently be used thereafter over it.
- For fines reduction during blasting in coal seam.
- For plugging the blasthole over the column explosives top so that the drill cuttings, (in case of iron ore) during stemming does not contaminate the explosives.

Question 135 : Write short notes on **Blasting Mats**.

Answer :

Blasting Mats

Blasting mats are generally made of steel cable or rubber tyre to

prevent the rock from flying when blasted. The size of the steel cable mats are generally 3m x 3m with a weight more than 1 tonne. After charging is over all the blastholes are covered with one or more mats and then blasted. Because of heavy in weight and holes are thoroughly covered the blasting mats prevent movement of flyrock. Since steel cable mats are lighter in weight compared to the rubber tyred mat and flexible in nature, they are the most popular in deep hole blasting. However in shallow holes, use of rubber tyre mats are advisable During laying the blasting mats the following points are to be kept in mind :

- The weight of the blasting mat over the blast should be optimum otherwise blast will throw the mat away, causing much of damage and allow enough flyrock.
- Blasting mat should never be dragged but placed smoothly over the blast taking due precaution to prevent cutting off or damaging the blasting circuits connected by the wires. No bare wire of blasting circuits should come in contact with the steel cables of the blasting mat, which may cause either short circuiting or partial detonation.
- While using blasting mats in shallow holes first of all old tyre is to be placed over the blast and then steel cable mat shall be placed over it, since in this case there exist much higher probabilities of ejection of explosives along with stemming material which may damage the mats.

Question 136 : Discuss elaborately the deep hole blasting practices in a surface mine.

Answer :

Deep hole Blasting

Before deciding to implement the deep hole blasting practices in a surface mine elaborate arrangement of planning, face preparation, decision of drilling and blasting pattern based on experience, cost economic analysis, safety, recording of the results, etc. are to be decided primarily in depth.

The mining block where the deep hole blasting is to be practised is to be levelled properly by a bulldozer. Particular care is to be taken to doze the front row to achieve a very good blast. A surveyor is to be engaged to demarcate the block area where the deep hole blasting is to be

practised. He should give particular attention to mark the front and the last row of drilling line and with the help of blasting overman he will arrange to mark the pattern of holes depending upon the site specific conditions and the rquirement of blast e.g., whether the purpose is for normal blasting or for side casting. A blast design plan and also a hand plan is to be prepared where most of the parameters of drilling and blasting are to be shown and are to be handed over to the drilling and blasting personnels. The experienced blasting engineer, foreman and drilling and blasting crew should be engaged in this purpose who will take decisions based on their experiences in the site itself about the changing or variation of drilling and blasting parameters based on the site specific conditions for achieving a very good blasting result. So there must be enough flexibility regarding the drilling pattern, charging of holes, firing pattern, negotiation of very difficult situations depending upon the prevailing conditions in the site. Here blasting engineer alongwith the blasting supervisors play key roles for making a decision.

The diameter and the depth of the blast holes are generally kept constant depending upon the size of the equipment. In case of blasting in a shovel face the height of the bench for 4.6 m^3 shovel is kept 10 to 12m, for 10m^3 shovel 14m and for 20m^3 shovel 20m. If the height of the benches become more during operational process they should be converted into an optimum height so that shovel boom can reach near to that point. Parameters are also to be kept in mind regarding caving of big boulders from the top edges of benches which should be minimum. The depth of the hole in case of a dragline face may be 30m or more for a dragline having a bucket capacity of 24m^3 and boom length 96 m.

The variable parameters in the deep hole blastings are burden, spacing, amount of charge, firing pattern, etc.

The spacing of holes should be more than the burden. In rare cases they are kept equal to the burden. If the burden is kept very small then there will be too much problems of flying rocks, excessive throw and also the amount of fine formations will be excessive. These characteristics, are very much undesirable in the shovel face. However, in the dragline face during side casting the first 4 to 5 front row holes burden are kept a little bit less for getting a good throw. The spacing is also kept less at the front row holes. During side casting with the deep hole blasting in 8 benches of 30 m height and at an angle of slope around

5^0 the front row spacing is kept nearly 6 m with a burden about 6 m but the spacings of next four row holes are about 9 m and burden are 6 m. The remaining holes are to be drilled in the normal manner [8 m burden x 10 m spacing (approx.)] (Fig. 55). The stemming height in the front row is about 9 m whereas the stemming height at the other rows varies from 6.5 m to 7 m. If the burden is kept very large then there will be the problems of poor fragmentation with the formation of huge amount of big boulders, problems of severe back breaks, high level of vibration, toe problems and also the problems of excessive fly rocks. In that case the shovel or other excavator will be unable to excavate those big boulders and the boulders will be required to be blasted by the secondary blasting technique. It will cause huge amount of delays since maximum time will be lost only to deal with the problems. Low production and productivity, high cost of production, less safety, etc. are also some of the major demerits worth to mention in this regard.

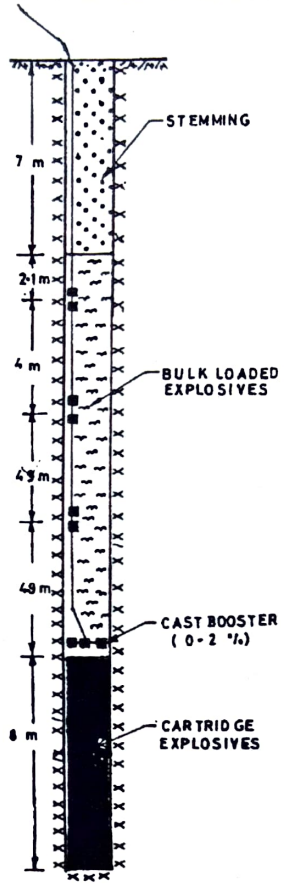


Fig. 55 The distribution of various charges in the deep blasthole

During drilling deep holes the problems like excesive back break zone, uneven drilling face, non-availability of the site for drilling, incorrect burden and spacing depending upon the local strata condition, incompetency and lack of technical knowledge of charging and firing the holes, improper firing sequence etc. are most common which will adversely affect the blasting efficiency and result.

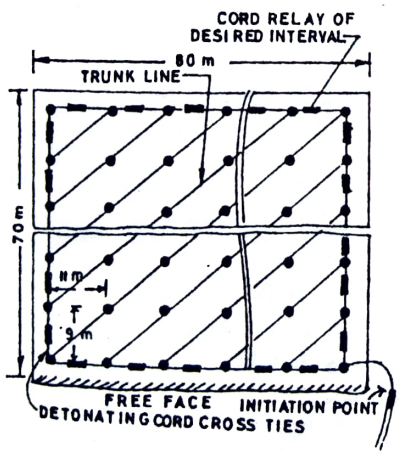


Fig. 56 Deep Hole Blasting with rectangular drilling pattern and diagonal firing system

In shovel face the staggered pattern of drilling, 'V' or half moon pattern of firing are very common for a larger length of firing. But in smaller area square pattern of drilling and diagonal firing pattern will yield the best result [Fig. 56].

If sufficient mineral or coal free excavated areas and sufficient free face are available and the size of the blast are also heavy then in the dragline face the length of the blasting zone is around 1.5 to 3 times the width of the cut by a dragline. That means if the width of the cut by a dragline is 65 m, the length of blast with an average hole depth of 30 m will be around 100 m to 200 m. But during same practice, it may be found that the length of the blasting block is less than the width of the cut by a dragline and also the free face is covered with the overburden dump and it is very near to the blasting site.

Subgrade drilling is necessary to ensure the breakage of toe of the bench at a satisfactory level. To clear the toe effectively the holes at the front rows should be spaced half compared to the other holes in the other rows and are to be charged alternate holes with half and full charges.

To combat side and back breaks and also to control the fly rocks the four side rows are to be stemmed 1.5 to 2 m more in height. Higher stemmings are also necessary in the front row and the last row to mobilize the explosive energy in the downward direction and also prevent the energy loosing out of the hole without doing any work. In the dragline bench with deep hole blasting, rectangular pattern of drilling with diagonal sequence of firing is followed. In watery holes since bulk explosive system does not give efficient result, 1/3rd of the explosive in each hole are of the cartridge type at the bottom. Ratio of primer explosive and non-primer explosive at the bottorn is 1:1.

For side casting in line firing using 50 mill-second delay detonator in the first 3 to 4 rows, and 75 and 100 millisecond delay detonators in the rest of the rows will give a very good fragmentation result [Fig. 57].

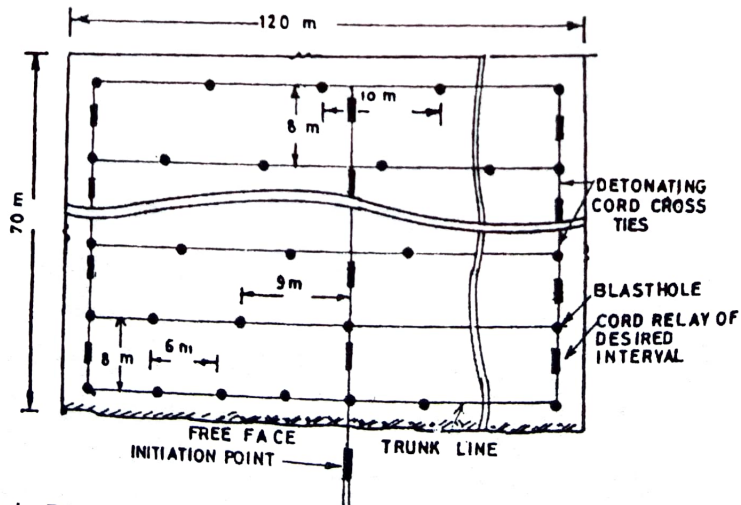


Fig. 57 Deep Hole Blasting for a side cast blast with triangular system of drilling and line firing system

"Fly rock" is the most important reason for endangering the safety of both men and machines. It can be properly controlled by optimum charging and stemming of the holes, using proper delay interval and keeping the burden distance optimum. In every blasting operation "Blast Plan" prepared by the surveyor showing the location, spacing and burden of each hole in the bench, loading sheet prepared by the supplier of the explosive in consultation with the blasting foreman or overman showing quantity of different kinds of explosives in each bole, "Blast Report" for studying the comparative performance of each blast in each section, "Performance Report of Explosives", "Blast Delay Report" which mention the reason of delays, "Blast Damage Report" which mention the frequency of damages of equipment due to blasting and the statutory "Shot Firing Report" prepared by the blast foreman or overman are very important.

Question 137 : Discuss ground vibration produced by blasting. Categorise the blasting damages due to vibration. What is the permissible peak particle velocity in India? How is ground vibration monitored? What are the the threshold limit values of sound vibration in abroad?

Answer :

Blast Vibration

Higher vibration due to blasting can damage the solid rock-mass by initiating fresh fractures or extending and dilating existing fractures. The vibration produced by the explosive decays as it propagated from the borehole due to work done in fragmentation, heat generation by an elastic process and geometric spreading. Ultimately the stress becomes below the elastic limit of the rock and passes through the rock as an elastic wave. The waves are reflected and refracted as they pass through the rock and the pulse is lengthened into a wave train which may be several seconds long for a single borehole at a distance of a few hundred metres. Depending on the geo-mechanical parameters and delay sequence various blast induced frequencies may be generated and some of them may be trapped in the strata layers. These trapped wave will resonate at frequencies prompted by the thickness and type of material at the surface. In surface mining operation these waves may cause damage of the surface structure near to the opencast working by two ways—

a) high amplitudes : it may force a structure to move/damage;

b) resonance : it transforms a low level vibration into one which may be annoying or damaging.

If the frequencies of the ground vibration match that of a structure, the vibration will be amplified and much more severe damaging effect may take place.

Elements of buildings like spring floor, stud partition walls, ceiling and windows react as man-spring system, each has it's own natural frequencies varying from 4-25 Hz. When the low frequency ground vibration coincides with the natural frequencies of the structure, resonance is produced. Under this resonance condition structure absorbs most energy progressively, orients towards deformation till the plastic deformation takes place. Hence even a low PPV of ground vibrations at natural frequency of structure is very harmful for the latter. Natural frequencies of brick and concrete structure generally varies from 8-16 Hz. The PPV is considered to be the best criterion for evaluating blast vibrations in terms of it's potential to cause damage. The blasting damage may be classified into the following 4 categories :

Category	**Damage**
I	No noticeable cracks/damage.
II	It comes under the threshold limit value. Symptoms—fall of plaster, extension of old cracks, dislodging of loose objects, loosening of joints, etc.
III	Minor damage. Symptoms—development of hair cracks in the masonary, fall of loose objects, mortar, broken windows, etc.
IV	Major damage. Symptoms—development of large cracks, serious weakening of structures, fall of masonary, cracking in the foundation, etc.

Maximum PPV which can be withstood by the rock before tensile failure (assuming a brittle failure mode of rock) can be expressed as follows :

$$PPV = \frac{\sigma_t \cdot V_p}{E}$$

Where

σ_t = tensile strength of the rock

Vp = P-wave velocity of propagation

E = Young's modulus of elasticity.

The PPV at a distance x from a blasthole containing weight 'W' explosive charge can be expressed as follows :

$$PPV = KX^{-\alpha} W^{\beta}$$

Where K, α and β are the site specific constants.

The USBM equation in this regards is as follows:

$$PPV = 1143\left(\frac{X}{W^{0.5}}\right)^{-1.6}$$

Where X = the distance of point of measurement from the blasthole, m.

W = weight of explosive detonating per delay, Kg.

Table-32 and 33 depicts the threshold limit values of ground vibration in India and abroad respectively.

Table 32

Permissible Peak Particle Velocity in India. (in mm/sec)
(At the foundation level of structure in the mining site)

Building/Structures	Frequency range		
	< 8 HZ	8-25 Hz	>25 Hz
Belonging to Owner			
i) Domestic houses/structures (made of brick, cement, mud, and also other temporary type structure)	10	15	25
ii) Industrial buildings/structure (made of RCC, and framed structures)	15	25	50
Not belonging to owner			
i) Domestic houses/structures (made of brick, cement, mud, and also other temporary type structure)	5	10	15
ii) Industrial buildings/structures (made of RCC, and framed structures)	10	20	25
iii) Objects of historical importance and sensitive structures	2	5	10

Table 33

Threshold Limit Value of Ground Vibration in Abroad

Table 33 (A)

FRANCE

Type of Structures	Peak particle velocity (mm/s)		
	4-8 HZ	8-30 Hz	30-100 Hz
Residential	8	12	15
Sensitive	6	9	12
Very sensitive	4	6	9

Table 33 (B)

GERMAN

Type of Structures	Peak particle velocity (mm/s)		
	<10 HZ	10-50 Hz	50-100 Hz
Commercial	20	20-40	40-50
Residential	5	5-15	15-20
Sensitive	3	3-8	8-10

Table 33 (C)

SWEEDEN

Subsoil	Vibration (mm/s)
Unsonsolidated strata of moraine sand, gravel, clay, etc.	18
Consolidated strata of moraine slate, soft lime stone, etc.	35
Granite, gneiss, hard limestone, quartzitic sandstone, etc.	70

Table 33 (D)

AUSTRALIAN STANDARD

Sl. No.	Type of Structure	PPV of the Ground (mm/sec)
1.	Building of historical importances, monuments special buildings	2
2.	Resisdential houses (low rise), Commercial buildings (ordinary type)	10
3.	Industrial and Commercial buildings, structure of RCC or steel construction	25

Table 33 (E)

USBM STANDARD

Sl. No.	Type of Structure	PPV of the Ground (mm/sec)	
		Low frequency (<40 Hz)	High frequency (>40 Hz)
1.	Modern houses, dry wall interior	19	50
2.	Older houses, plaster on wood, construction for interior walls	12.5	50

Table 33 (F)

USSR STANDARD

Sl. No.	Type of Structure	Allowable PPV of the Ground (mm/sec)	
		Repeated	One fold
1.	Hospital	8.0	30.0
2.	Children institutions, large panel residential buildings	15.0	30.0
3.	All types of residential buildings (except large panel), Office and industrial houses having deformations, boiler houses, high brick chimneys	30.0	60.0
4.	Office and industrial buildings, RCC pipes, traffic fly overs, railway tunnels, etc.	60.0	120.0
5.	Single story skeleton type industrial buildings, primary mine openings (service life upto 10 years), pit bottoms, main entries, drifts	120.0	240.0
6.	Secondary mine openings (service life upto 3 years), haulage roadways, drifts	240.0	480.0

Monitoring of Ground Vibration

It is measured by the self recording displacement seismograph or other portable microprocessor based ground vibration monitoring instruments, equipped with vibration transducer. The 2000 DK Portable Seismograph and SSU Micro Seismograph manufactured by the M/s Geosonics Inc., USA can monitor the ground vibration in the seismic trigger range varying from 0.6 mm/s to maximum 254 mm/s and air

trigger range varying from 106 dB to 140 dB. It has printer and disc recording facilities. The instrument can be used to determine the peak particle and resultant velocity, peak acceleration and displacement, air overpressure and half wave frequency, etc.

The triaxial transducers are also used with other type blast monitoring instrument. The transducers are placed over the solid undisturbed ground near to the structure. For structural response the transducers are placed horizontally over the wall, floor and ceillings. A minimum of 15 points of observations at different places corresponding to a minimum of 10 blasts shall be made for better results. The least mean square method of regression analysis shall be used to interpret the data. When both the blasting and measurements are done either on the surface or in the underground the square root scale is chosen but when blasting is done in the underground and measurement is done on the surface, the cube root scaled distance is chosen.

Question 138 : Chalk out some salient suggestions for controlled deep hole blasting in an opencast coal mine.

Answer :

Suggestions for controlled Deep Hole Blasting in an Opencast Coal Mine

1. The working should be done with the controlled blasting technique with a maximum charge per delay not exceeding 250 kgs.
2. The pattern of holes and charging details in the medium hard sandstone rock to be followed is given below :

Bench Height (m)	Spacing X Burden (m)	Charge/hole (kgs)	Stemming Column
7.0 to 10.0	5 x 5	40 to 65	Not less than 5 m

3. A charge ratio of about 4 m /kg shall be used in medium hard sand stone.
4. Moist sand (should be tramped hard into the shot holes) shall be used as stemming material in the overburden benches.
5. Any conventional type of explosives and ANFO can be used.
6. Short delay detonators of 5 m.s. type should be used.
7. Depth of each hole should be measured before charging.
8. A long reel of shot firing cable should be provided.
9. The code of signals as prescribed by manager shall strictly be followed.

10. The colliery manager shall from time to time satisfy himself that all the above recommendations are strictly followed and should maintain a detailed record of each blast on the lines as in the report and should be countersigned for his confirmation.
11. While approaching a distance of 500 m from a very important structure further study for monitoring the ground vibration should be done. The working should not be extended beyond 150 m from that important structure.
12. One drilling and blasting engineer should be engaged exclusively for the supervision of the drilling and blasting job.

In this connection some of the following salient points are worth to mention :

1. If any permanent building or structure of permanent nature, not belonging to the owner lies within the danger zone of an opencast working, the aggregate maximum charge in all the holes fired at one time shall not exceed 2 kg unless permitted in writing by the Chief Inspector or the Regional Inspector and sujbect to such condition as he may specify therein. Provided that if blasting is done with delay detonator or other means and that there is a delay of atleast half second between successive shots fired, a maximum charge of 2 kg could be used in each hole.

   Provided further that if the shots distance from the place of firing to any part of such buildings or structure is less than 50 m irrespective of the amount of charge, no blasting shall be done except with the permission in writing of the Chief Inspector or the Regional Inspector and subject to such conditions as he may specify therein.
2. Generally for this purpose controlled blasting technique is enforced. With the adoption of controlled blasting it is possible to control overbreaks, ground vibration, control of uneven and rough perimeter of face and also possible to create a smooth wall. The technique helps to stabilise the pit slope, and improve the economy of blasting. It can be done in the following manner

a) Presplitting—already discussed
b) Cushion blasting – already discussed

c) Smooth wall blasting
d) Line drilling
e) Buffer blasting

Line Drilling

In this method holes are drilled along the contour line at a close spacing and left uncharged. It will create a plane of weakness to which last row of production blast (generally they are lightly charged) will break. It is very difficult to maintain the accuracy of drilling and cost involved in this technique is also very high. In this system a greater amount of drilling is required and it is applied only to homogeneous rock. The distance between these unloaded rows which are closely spaced (0.7 to 0.8 times the normal spacing and charged with half the quantity of explosive evenly distributed by deck charging compared to the normal blastholes) is approximately 0.4 to 0.6 times the burden of the production hole. This distance is termed as 'Buffer distance'. The holes in the line drilling system help to reflect the shock wave and help to reduce the shattering effect of the rock outside the periphery. Generally the diameter of the holes are 50 to 75 mm and spacing the holes is generally two times the diameter of the holes depending upon the hardness and formation of rock (in homogeneous rock more spacing where as in fissured and closely jointed rock less spacing is desirable). This type of practice shall not be done in inhomogeneous rock since plane of weakness in this case may extend in the finished wall. The line drilling will give result in homogeneous rock and in benches having smaller width.

Smooth Wall Blasting

This technique is generally used in the underground and construction works and it is quite similar to the cushion blasting. It is nicely applicable to homogeneous rocks. In this case row of horizontal holes at the perimeter are blasted after blasting the lifters using the last delay number in the firing sequence. Holes at the perimeter are drilled closely and charged with low density decoupled cartridges. After charging the holes they are nicely stemmed upto the collar of the holes and all the holes are fired simultaneously. For good shearing action it would be better if lower rib holes are blasted after clearing the muck pile; otherwise it will yield poor result because of the confinement by the

muck pile. Generally the burden of the smooth wall blasting holes are 1.4 to 1.5 times the spacing between the holes. This technique reduces the generation of cracks and overbreak in the side walls and on the roof, improves finishing condition of the excavation, reduces much using of supporting material and hence saves cost of excavation and cost of it's maintenance.

Question 138 (A) : Discuss briefly EXEL-NTD surface delay units and EXEL-LP series detonators.

Answer : EXEL-NTD surface delay unit are a non electric millisecond delay detonator for surface initiation of EXEL downline detonators whereas EXEL-LP series detonators are a non-electric long period delay detonator for initiation to explosive charge in the blast holes. Both the EXEL-NTD and EXEL-LP detonators consist of a non-electric detonator, a length of shock tube and a cobra connector. The non-electric detonator is a precise full strength unit, with a predetermined delay element inside and the shock tube is a high strength, high abrasion resistant tubing which transmits the initiation signal to the detonator. One end of the shock tube is crimped into the detonator shell and a water-proof ultrasonic seal closes off the other end.

In case of EXEL-LP series detonator delay number is printed on the J. Hook attached to the signal tube towards its crimped end, whereas in case of EXEL-NTD surface delay detonator unit is housed in colour coded bunch block connector for easy identification in the field. The main component of a non-electric system is the shock tube. This is a hellow tube made with advanced materials designed to withstand field conditions. The tube comprises of 2 layers, the inner layer made of special material 'surlyn' coated with a very fine layer of a reactive mixture. The reactive mixture (HMX and Aluminium) used is in the range of 14 to 16 mg/m of tube. This coating is done at the time of the tube extrusion and thereby good adhesion is ensured. The outer layer is a LLDPE material and is designed to withstand the stresses from day to day use in terms of abrasion and tension. When an initiation shock signal is introduced at one end of this tube, the reactive mixture propagates the signal all along the tube even through knots and kinks, thus making it a viable mean of carrying a blast initiation signal in downlines as well as trunk-lines. The initial signal may be introduced either by a simple blasting cap or by a non-electric detonator, or by a

plain detonator or a detonating cord as the case may be.

Both the EXEL-NTD and EXEL-LP series detonators provides a high level of safety against initiation by static electricity, stray electrical currents and radio frequency transmission. EXEL-LP series detonators used inside blstholes should always be secured inside a suitable primer which fully encloses the detonator shell to protect it from abraison or damage during charging. Bare (exposed) detonators should not be placed inside the blastholes. The following are it's delay timings—

LP delay Delay	1	2	3	4	5	6	7	8	9	10
periods (ms) :	200	400	600	1000	1400	1800	2400	3000	3800	4600

EXEL-NTD surface delay units has the following colours—

Delay period (ms)	17	25	42	65
Bunch block colour :	Yellow	Red	White	Black

Question 139 : With the help of an example discuss elaborately modern environmental friendly blasting technology using shock tube detonator.

Answer : The initiation system used in the blast contributes a lot to the blast design. Presently special attention is being paid on selection of explosive, blasting accessories vis-a-vis rock characteristics with the objective of attaining optimum fragmentation of huge volumes of over-burden materials from the opencast mines. The major innovation in the area of blasting accessories in the present century is the non-electric shock tube detonators. Shock tube detonatiors like RAYDET from IDL chemicals, EXEL from ICI, premier shock tube detonators from M/s. Primer Explosives Ltd. etc. revolutionise the system of opencast blasting technology after offering several advantages like drastic reduction of noise and vibration, controlling of fly rock problems, etc. This shock tube detonators technology is definitely a boon of blasting technology proximity to the human habitations around mine sites.

Shock Tube Detonators

EXEL of M/s ICI Ltd.

It is a plastic tube of 3 mm to 4 mm outer and 1.5 mm inner diameter coating inside with an explosive composition. During manufacturing extrusion process HMX (high melting explosive) and aluminium powder mixture is sprayed at a quantity of 20 mg/m length of

the tube. The shock tube is sealed at one end has a delay detonator crimped to the other end. The EXEL uses common plastics and suitable additives which make it sticky during molten stage so that the explosives power sticks properly to its inner wall. It's melting point is 200^0 C and tensile strength is more than 45 kg. When sealed end of the tube is fired with detonator or detonating cord, a mild detonator wave travels through the tube at a velocity around 2000 m/s and after reaching to the other end it initiates the detonator which in turn fire the primer attached with it.

RAYDET of IDL Chemicals

It is a non-electric delay initiating device. It consists of a polymer tube with a very small quantity of explosive material on its surface. Raydet tube has 3 mm outer and 1.5 mm inner diameter and amount of explosive present in the inner annular space of the tube is 10 to 15 mg/m length of the tube. A no. 8 strength instantaneous or any one of the 20 delay periods in the millisecond range or any one of the 12 delay period of half second range (currently available) is crimped to one end of the tube for initiation. The other end is sealed to prevent ingress of moisture and other foreign matter. Raydet can be initiated by a detonator or detonating cord. Upon initiation a low order shock wave travels through the tube and reliably initiate the detonator. The RAYDET DTH (down the hole) is used in hole in conjuction with surface unit RAYDET TLD (Trunk line delay). Generally Raydet DTH has 475 ms, 500 ms, etc. delays whereas Raydet TLD consists of 0, 17, 25, 34, 42, 50, 67 and 100 ms delay with different assigned colours. Raydet tube lengths are available in various sizes but the maximum size is 45m.

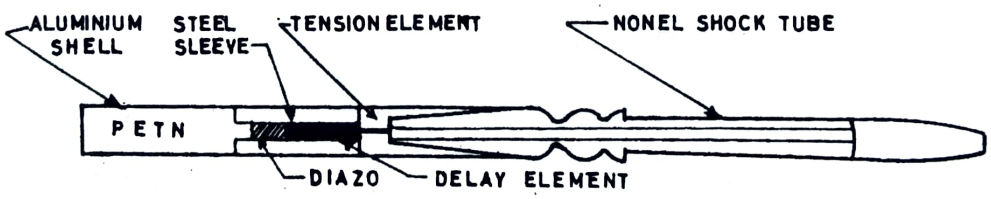


Fig. 58 A schematic line Diagram of a shock tube detonator

Merits and Demerits of Shock Tube Detonators (Fig. 58)

Merits

The shock tube detorators offer the following merits :

- Assists efficient bottom firing
- Offer higher degree of safety

- Heavily reduce noise, vibration and flyrock
- Does not cause desensitisation of explosive
- Combinations of delay in infinite number is possible
- Probability of cut-offs is practically nil
- It's handling and uses are very simple
- Initiation can be done from bottom-up
- Degree of fragmentation and profile of muck pile are excellent by the use of shock tube detonators.
- Cratering affect is nil.
- It is immune to extraneous electricity sources, flame, fire and stray current.
- It is insensitive to friction and impact effects.
- It gives an output which is excellent initiator of polytechnic compositions.
- It is highly reliable.

Demerits

However it has certain demerits too which are as follows :

- It has poor ultraviolet resistance - Sun rays cause degradation of the shock tubes. However EXEL has the better Ultraviolet protection.
- It has low tensile strength. Tensile strength of EXEL is of course more than 45 Kg due to use of surlyn.
- It has poor abraison resistance. However EXEL has higher abraison resistant characteristics.
- It has poor oil resistant characteristics.
- Higher cost.

Surface connection

Surface hook up with the blast hole shock tubes should be done with the shock tubes itself and it is commonly done by the EXEL LTD (trunk line delay) detonators. The EXEL LTD detonator comprises a specific length of tube crimped to a delay detonator which is enclosed in a specially made plastic block. Each TLD block can hold and initiate upto 5 EXEL tubes. The surface hook up can also be done by the EXEL Handidet.

Emulsion Explosive

The basic ingredients of emulsion explosives include inorganic nitrates, fuel oil, water, emulsifier, wax and atomised aluminium. They are basically water-in-oil type emulsions in which the aqueous oxidiser solution consitutes internal phase and the emulsifier concentrates at oil-water interface providing stability due to its film forming capability. The sensitization of these emulsions is achieved by the addition of micro-balloons, perilite or by chemical gassing of products. In case of softer strata condition the excess energy of the emulsion explosive is wasted and hence doped emulsions are used under such condition. Doped emulsions are prepared by adding 10% to 20% of Ammonium Nitrates prills with 80% to 90% of strength emulsions. This doped emulsions have lower densities, higher energies and better sensitivity at the bottom in deep holes and also it has excellent water resistance properties.

Emulsion explosives have high VOD (4000-5500 m/s) for generating high shattering power, excellent water resistance property, and offer a high degree of safety during manufacturing, transporting and using. They are also comparatively insensitive to accidental initiation by friction, stimuli and fire. The IDL's bulk emulsion explosives system consists of two parts viz satellite plant and pump-truck support tanker. The basic emulsion matrix is manufactured in the satellite plant whereas the pump truck is desired to produce either EMULKING straight emulsion explosive or SUPERMIX emulsion blended ANFO.

Salient points on initiation

During bottom-upward initiation they reduce the risk of ground movement cut off. Rock movement due to detonation of the first row may cause slippage along the weak plane and hence cut-off the downlines in the 2nd row. If the downlines are made up of detonating cord this would result in a misfire. However if shock tube downlines are used then the delay detonators at the bottom would fire and set-off the explosive column. Generally explosives column is initiated first at the bottom of the hole causing explosive energy to work longest period at the bottom which results in significantly reduction in toe problems. But the most vital point is that all the shock tube detonators receive fire signal at the same time. This property makes it possible to combine surface at DTH delays, thereby enabling a series of infinite delay combinations. This leads to the possibility of firing a large number of holes with many delays

without any cut-offs and under limited noise and vibration control. Detonating cord trunkline produces noise but the use of TLD is practically noiseless and also bottom priming leads to significant reduction in the production of flyrock compared to the top priming. Beside this the use of shock tube downline eliminates disturbances to the explosive column and therefore there is no desensitisation.

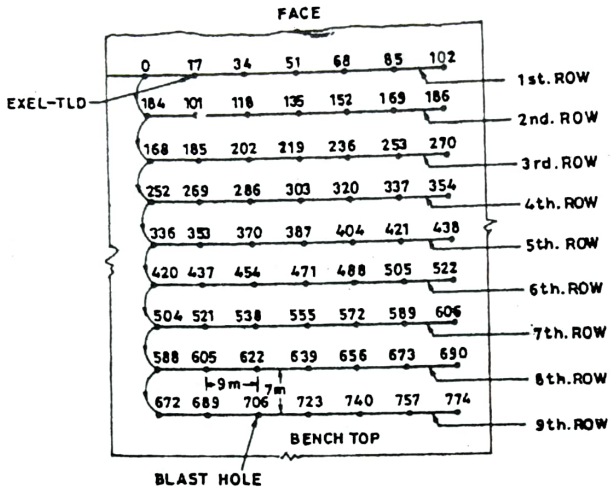


Fig. 59 Drilling and Blasting pattern in a dragline bench (all the digits mentioned are the delay sequence in ms)

A Case Study

Fig. 59 and 60 show the drilling and blasting pattern of a deep hole blasting in a dragline bench in one of the opencast mines of the Mahanadi Coalfield Ltd. The following are the drilling and blasting parameters in brief :

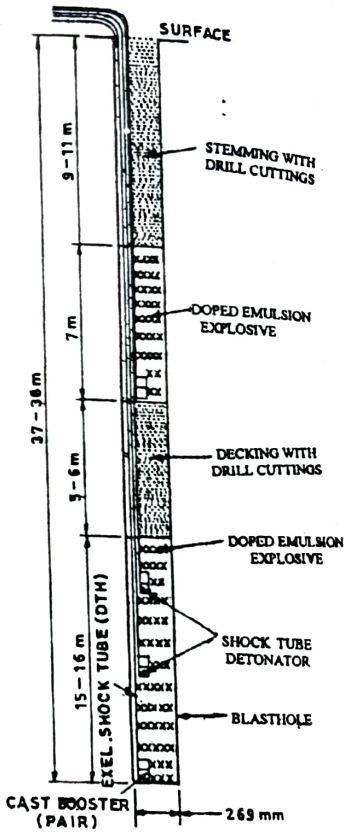


Fig. 60 Cross section of a blast hole in a dragline bench

Bench width	: >60 m
Average bench height	: 37 m
Average depth of blast hole	: 38 m
Diameter of blast hole	: 269 mm
Average spacing of blast hole in a row	: 9 m
Average burden between the two rows	: 7 m
Average number of holes per row	: 7
Average number of rows per blast	: 9
Explosive used	: Powergel

	(Powergel is a doped emulsion which is plant mixed, transported by the bulk delivery system trucks and at the sites it is doped with prilled ammonium nitrate).
Average amount of explosive per hole	: 1.2 tonne
Primer used	: Cast booster
Amount of cast booster (max.)	: 2% of the total weight of emulsion explosive.
Deck/Non-deck charging	: Deck charging employed (Fig. 60)
Stemming material used	: Drill cuttings.
No. of cast booster used per EXEL shock tube detonator	: 2 (combined weight : 500 gms)
Pair of cast booster used in the explosive column	: One pair / 5 m
Delay of EXEL - DTH	: 400 ms, 425 ms. 450 ms, 475 ms, etc.
Delay of EXEL - TLD	: O ms, 17 ms, 34 ms, 51 ms etc. (shown in the fig. 59)

Special plastic connectors and plastic blocks are being used for connecting the shock tubes.

From the figure-59 it is very clear that before initiating complete TLD detonators and a single DTH detonator in the first row shock energy transmits to a number of rows (5 to 6 rows) in the out-bye side of the face and hence cut-off both in downline and trunkline does not arise at all.

Result

The following results were obtained after the blasting :

- Excellent fragmentation and muck profile.
- Practically no noise and no vibration witnessed.
- No overbreak and ledge formation.

- No energy loss.
- No stemming ejection.
- Only little heaving and frontal rock movement observed.
- However blasting cost is more due to the use of costly EXEL shock tube detonators.

Question 140 : Write short note on 'Blasting Shelter'.

Answer :

Blasting Shelter

A blasting shelter is a temporary structure made of steel and is mainly constructed for the opencast working. However, for underground workings also a blasting shelter can be prepared, if needed. This is a very robust construction having a very strong roof, made specially for blaster and his assistants for taking shelter into it before firing a shot. Construction of such type of shiftable structures should be such that they can be shifted easily by dismantling. The skeleton of the structure is made either of steel pipes or channel girders or rails; and also the whole structure is covered by the strong steel sheets. No wooden roof or other materials which are liable to break easily due to the impact effects of projectiles should be used for their construction. For obtaining the cushioning effects sand bags are generally kept over the strong roof and also around the side walls of blasting shelters. Since the construction of blasting shelter depends upon many factors like the blasting parameters, surface topography, sheltering position, number of persons are to be sheltered, etc., specification of such type of blasting shelters will also vary accordingly. But whatever may be the conditions a blasting shelter should be substantially built taking due considerations of all the safety aspects. In opencast working it will be always better if the built up area is vacated beyond the danger zone i.e., a distance of 300 m. But if it can not be done, controlled blasting techniques with the help of milli-socond delay detonators and/or muffling of shots should be done in a manner that the blasting projectiles do not fly beyond a distance of 10m from the centre of the charge. The unsafe practices of taking shelters before firing shots like taking shelter under the old bucket of the dragline or shovel, taking shelter underneath the chasis of dumper, in the operator's cabin of the dragline, shovel, dumper etc., should be strictly avoided.

The shotfirer shall before a shot is charged, stemmed or fired, see

that all the persons other than his assistants in the vicinity have taken proper shelter. He shall also take suitable steps to prevent any person approaching the shot and he himself along with his assistants take adequate shelter. The blaster should also give sufficient warnings by the efficient system of signals or by other means approved by the manager over the entire area of the danger zone.

Question 141 : Chalk out some salient suggestions for extraction of developed coal pillars by the opencast methods.

Answer : Suggestions for extraction of Developed Pillars by the Opencast Methods :

1. Proper isolation by explosion-proof stoppings should be made in the smaller panel in case the workings are accessible before start of extraction of pillars by opencast method.
2. Position of underground galleries and junctions should be preferably marked on the overburden benches during extraction, through re-surveying and proper correlation.
3. Extraction of coal should be as fast as possible preferably within the incubation period.
4. So that heavy earth moving machines do not bog down the blasting pattern should be such that it packs the developed galleries in advance.
5. During mining due consideration should be taken regarding the position of developed galleries and likely fire in the galleries.
6. If access is available stone dusting and guniting may also be done in the predeveloped galleries long before the blasting operation is practised. There should not be the presence of any worker in that underground mine in that seam/section/other seam connected therewith.
7. If the holes are punctured into the old developed galleries, then the bottom of the holes are to be plugged with the help of wooden cones (splitted into two parts) or by power plus. Stemming material like wet or water saturated coarse rock chips, drill cuttings or sand are then to be dropped into the holes in multideck (to prevent release of full energy through the bottom portion of the holes), then the cartridge of slurry or

emulsion explosive using slurry booster cartridge over it. Again stemming have to be done into the holes upto a required length and then only a limited number of holes (10 to 12 numbers) are to be blasted (Fig. 61),

8. Special precaution may have to be taken during blasting operation.
9. A fire fighting plan must be chalked out to combat fire.
10. Arrangement of fire fighting with water jets with enough pressure should always be kept ready.
11. Quality control measures have also to be planned in the method of mining.
12. No shotholes should be drilled immediately above the galleries. Drilling and blasting operation should only be done in coal pillars.
13. If there is any multisection workings in the same coal seam in the same developed pillars then the bottom of the drilled shotholes should be atleast 3 m above the working of the lower section. In every round of blasting the thickness of the above parting can be determined by drilling a pilot hole in that section. A pilot hole may also be drilled to determine the thickness of parting (which should be atleast 3 m) between the top of the coal seam and the bottom of the shothole in the overburden bench immediately above the coal seam.
14. Bottom of all the shotholes in the overburden or coal benches should nicely be stemmed with moist sand at a length varying from 0.8 m to 1 m.

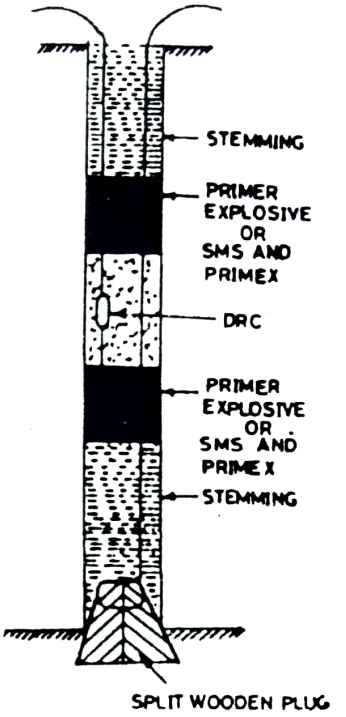


Fig. 61 Charging of punctured hole with multi deck in hole method.

Question 142 : What special steps are generally to be taken to blast hot holes in opencast coal mines? Discuss.

Answer : Special steps are generally taken to blast hot holes in opencast mines which are as following :

1. Measure the temperature of the hot holes with the help of reliable instruments.
2. Flush the hot holes with water at least 12 hours before blasting.
3. Plug the mouth of the flushed holes with the help of wooden plug to stop circulation of air through the holes to prevent heating of the same.
4. Measure the temperature of the holes with the help of reliable instruements at regular intervals.
5. If the holes do not retain water flush them with bentonite, sodium silicate and water mixture till water retains in the holes.
6. If cracks are found in the holes, flush them with water-guar gum (upto 5%) mixture to fill these cracks.
7. Cut the detonating fuse in proper length and pass them through the centre of each charge cartridge heat resistant emulsion or site mixed slurrry explosives.
8. When the temperature of the hole comes down to 80^0 C and charge the above mentioned holes with the heat resistant emulsion or slurry explosives. Nitro Glycerine based explosive and LOX are prohibited to use in the hot holes. Charging operation is to be completed within 2 hours.
9. Stem the holes, with wet or water saturated coarse rock chips, drill cuttings or sand.
10. Blast 10 to 12 number holes at a time.
11. If the temperature of the holes is above 80^0 C lowering down of the temperature is not further possible, then asbestos cement pipes are to be lowered in the holes (maximum 4 in number) and bentonite, sodium silicate and water mixture is to be flushed through the annulus between the pipe and the hole. Heat resistant emulsion or slurry explosives are to be lowered inside the pipe and then empty space is to be filled up with water, bentonite and sodium silicate mixture over which stemming is done with rock chips, drill cuttings or sand. The holes are to be blasted within one hour.
12. Holes which have temperature more than 100^0C generally

should not be blasted since at above 100°C the chances of misfires of fuses increase.

13. The temperature of holes can be measured by embedding Pt-100 temperature sensors with the explosive cartridge.
14. If the holes are punctured into the old developed galleries, then the bottom of the holes are to be plugged with the help of wooden cones (splitted into two parts) a by power plug. Stemming material as mentioned above are then dropped into the holes upto a height of 2 to 3 m. Charge the holes in multideck (to prevent release of full energy through the bottom portion of the holes) with cartridge of slurry or emulsion explosives using slurry booster cartridge. Cast booster should not be used. Again after stemming the holes to a required height a limited number of holes are to be blasted (Fig. 61). If the holes are charged without any decking arrangement use of the top priming is the best proposition.

Question 143 : Give an example of hot hole blasting practices in an opencast coal mine.

Answer :

Hot Hole Blasting in the Laxmi Opencast Mine

The following measures are to be taken before blasting into a hot hole :

- measurement of temperature at the top, middle and bottom of the blast-hole immediately after the drilling is completed.
- determination of rate of rise of temperature at the blast-hole at a predetermined sequence at the time span of 6 hours after the blast-hole drilling.
- determination of the effectiveness of cooling of the borehole temperature every after 6 hours after circulation of water.

The pertinent details of the hot hole blasting in this mine is as follows :

Active fire in the seams	: XIIA, XII and XI/XII. The seams are already developed.
Dip of seam	: 1 in 10
Burden (m)	: 2

Spacing (m) : 2.8

Diameter of blast-hole (mm) : 150

Quenching methodology : It was revealed that the equilibrium temperature value reaches within 6 hours after drilling. Some time it used to reach a value of 120^0 C within 6 hours after quenching and after 12 hours a significant amount of temperature used to rise. The blast holes used to be cool down by water flushing 12 hours before charging with explosives to lower down the temperature below 80^0 C. The water used to be pumped into the holes from the nearby jora by a pump.

During flushing water into the hot holes care was taken regarding the retaining of water into them. If the water does not retain there a mixture of bentonite, sodium silicate and water used to flush into the hot-holes since mixture ratains water and maintain the temperature. Gaur gum upto 5% mixed with water is also flushed sometimes into the hot holes to seal any existing cracks and fractures.

The complete cycle of charging and blasting was done within 2 hours to avoid rising of temperature into the blast-holes.

Number of holes blasted/round : 12

Explosive used : Slurry explosive (powergel-C and powergel-1)

Charge factor (kg/m^3) : 0.35 to 0.38

Exposed coal used to be blanketed by aluvium soil, morum, etc. to restrict the air in contact with the fire in the coal seam. Moreover the exposed coal is also flushed with huge quantity of water from the nearby Jore. Water is also used to be pumped out from the underground X seam workings by a submersible pump.

Equipment used

Hydraulic Shovel : 3 numbers, bucket capacity 2.5 m^3

Backhoe : 2 Numbers, bucket capacity 1.2 m^3

Haulpaks : 4 Numbers

Terex : 5 Numbers

Hoses and seals of the hydraulic excavators used to get damage frequently during dealing the blasted hot coal.

Question 144 : What measures are to be taken to blast hot holes in opencast coal mines during the following condition

a) If the temperature of the hole is in between 60^0C and 120^0C

b) If the holes are punctured into the old developed galleries.

Answer : Consult the answer to Question 142.

Question 145 : Mention some salient points regarding the safe practices in shot firing.

Answer :

1. To safeguard against 'kinking' which may cause misfires, do not coil safety fuse too tightly or twist the safety fuse coil or straighten the lead wire of the electric detonator carelessly in case of firing with safety fuse and electric shotfiring respectively. While straightening the lead wires of the electric detonator grip the wires about 10 cm away from the detonator with one hand and smoothen out the wires with the help of the other hand to avoid any pull on the fusehead.
2. In case of electric detonator, insert the detonator into the cartridge until it is completely buried. The hole in the cartridge for this purpose should be made with the pricker made of wood or other non-ferrous material. After inserting detonator a hitch is to be made with the lead wires around the cartridge to prevent accidental withdrawl of the detonator from the cartridge during loading. To guard against accumulation of any stray current detonator leads should be short-circuited.
3. In case of firing with the safety fuse the length of the fuse should be adequate (minimum 1.2 m) for allowing the shotfirer to reach a safety blasting shelter before detonating the charges. The end (5 to 6 cm) of the safety fuse should be squarely cut with a clean sharp knife for obtaining a fresh surface to insert the same end into the detonator. The end of the safety fuse should not be damped or come in contact with any oil or grease. After inserting the fuse into the detonator, crimp the later on the fuse with the help of a proper crimper, but never use teeth for crimping. During crimping fuse should be in contact with the detonator composition. Avoid forcing or screwing of the fuse into the detonator tube. The length of the crimp should be of 3.6 mm from the open end of the tube.

4. Never should fuse be kept lying exposed in the sun for a long period which will lead to increase the burning speed of the fuse.
5. When the charge is to be fired under watery conditions cap-sealing compound should be used to prevent penetration of moisture.
6. After inserting the detonator into the cartridge in the manner as already mentioned in the question no. 114 the cartridge wrapper should be tied up firmly around the fuse just above the detonator.
7. The primer cartridge should be inserted into the hole pointing the detonator towards the main charge.
8. Charging of bulk explosives should only be done by a trained person and only the minimum necessary number of persons should be present near to the bulk loading equipment while charging. Care should be taken while pumping to avoid spillage.
9. Drilling (which is done at the distinctly marked position) and charging of deep holes should not be carried out in the same area at the same time. A detailed record of the hole positions, type of explosives, quantity of explosives, hole depth, charge column and stemming should be maintained for finding out the depth of the charge in case of a misfire. Before firing the holes, all the holes should be rechecked regarding the proper connections and installation of delays in the proper firing sequence and also the blaster should be confirmed regarding the removal of the surplus explosives, persons, equipments, vehicles, etc., from the blasting site, The blaster should also confirm that the proper warnings are given in time with proper means and guards are posted with the red flags at a safe distance around the site to prevent persons approaching the danger area inadvertently while the shots are being fired. Before firing the shot holes, the shot firer should give a final warning and ensure that all persons within the danger zone taken proper shelter. In surface mines blasting operation should be completely stopped during an electrical storm and all persons shall leave the blasting site.

10. To guard against any sparking there should be no ferrous materials in the stemming. The first few centimetres of stemming should be pressed home gently thereafter pressure is to be applied gradually. A cartridge should not be forced to insert into the hole. If a cartridge gets jammed in a shot hole, nobody should attempt to dislodge it with a drill steel or any other metallic rod but treat it as a misfire. The stemming rod should be made of wood or some non-ferrous non-sparking material. During stemming hold the safety fuse to one side of the hole to prevent damage to the safety fuse or hold the lead wires of the electric detonator to one side of the hole to avoid any stripping of insulation in case of firing with safety fuse and electric shot firing respectively.
11. During firing shots blaster should be sure that strands of one conductor of twin core cables are not in contact with the strands of other conductor. This problem may be solved if the length of the exposed ends of the conductors are staggered in relation to each other. The firing circuit should be kept insulated from the ground, bare wires, rails, pipes or any other path of stray currents. For making connection bare ends of the conductors should be twisted together tightly for a length of about 3 cm. Water-proof insulating tape shall be used in watery condition for ensuring good insulation and to avoid any short-circuiting.
12. Before firing any shot all persons should be withdrawn from the blasting site to a safe place (in opencast mine this distance is more than 100 m). First of all with the help of an ohmmeter the insulation of the blasting cable is to be tested. The test is as following :
    a) Keep the two leads of the blasting cable at one end separated and connect the two leads of the other end of the cable to an ohmmeter. In this case no current should flow and the resistance of the cable should read infinity.
    b) Now join the separated conductor together to complete the circuit and check the continuity of the cable. Read the resistance of the cable.

Then to check the continuity of the blasting firing circuits it is

also to be tested by an ohmmeter from the firing point at a place of safety. If faulty circuit is detected the blaster/shotfirer should disconnect the blasting cable from the ohmmeter and examine the circuit for the faults. If it is detected then it shall be rectified and retested again with the ohmmeter. If the test signals the shotfiring circuit is healthy in all respects, shotholes are to be fired in the normal manner. If the fault cannot be detected in this manner, the circuit should be split into two halves. Test these two halves separately with the ohmmeter for continuity and identify the defective half. Again divide the defective half of the circuit into two halves and identify the defective quarter by repeating the test. Continue this testing operation till only 3 to 4 electric detonators remain in the selective portion of the circuit. Next each detonator should be tested to find out the fault. After locating the defective detonator in the said manner, the remaining detonators in the circuit should be connected in series and fired. The undetonated shot should be treated as a misfire and to be dealt with accordingly. However, in surface mines if the detonators are kept on the surface, the faulty detonator located in this manner should be replaced by a fresh detonator. The circuit thereafter is to be reconnected and tested by an ohmmeter and be fired normally.

Question 146 : How is dilution of ore controlled by drilling and blasting in opencast benches? What are the main problems generally faced during drilling in loosen rock? What advantages are generally obtained for blasting in good jointed rock? How are ultimate pit slopes formed?

Answer : Since bulk rock blasting results great losses and dilution of the mineral, selective rock blasting should generally be practised where different kinds of rocks are found in the benches. When upper rock layer make an angle nearly 20^0 to 30^0 to the horizontal with the lower layer, then upper layer is to be blasted first in rows after drilling holes to an appropriate length in the upper part of the benches. But it will result low production and productivity, high explosive consumption, drilling of higher meterage of holes, delay of sequence of operation of cycle and invlovement of huge manpower. However, when the layers are inclined at an angle of 60^0 to 80^0 then inclined boreholes are drilled to separate out

different kinds of rock from the bench. So in all the cases as mentioned the blasting block will be shorter in height and width. If the rocks with difference in nature and type exist in the mining block lengthwise then selective blasting in those patches/sections is to be practised lengthwise based on the occurance of mineral deposit. The dilution of ore may be controlled by selecting the suitable firing sequence. Separation of various grades of ore from the bench is possible by controlled blasting by casting ore from various parts of the benches into the muck-pile of adjacent part or in the periphery of the benches. In loosen rock the problems faced are caving of holes, deterioration of drilled blast holes and drilling condition, less meterage of drilling, etc. So for drilling a casing pipe is essential. The loosen layer may also form due to the overdrilling and blasting of holes in the overburden. If the rocks to be blasted have very good joints then bench height may be increased and the advantages of less amount of drilling meterage, reduced consumption of explosives, etc., may be gained. Blasting has to be done precisely to stabilise the ultimate pit slope near the pit limit to orient the benches to the ultimate position. This is possible by the following ways :

1) By practising diagonal pattern of short-delay firing of holes longitudinally starting from a distance of 40 to 50 m from the ultimate pit slope limit.
2) By forming a cut zone along the ultimate pit limit by drilling holes of more than the bench depth keeping them at 1 to 4 m apart and charging with 0.5 to 3 kg/m of explosive with the help of detonating cords.
3) By drilling and blasting of inclined (inclination should match the inclination of the bench slope) bore holes of 60 to 100 mm diameter spacing at 2 to 3 m apart. These will dislodge rock at the line of holes and they do not cause backbreak.

Question 147 : What general safety measures are to be taken for blasting practices?

Answer :

General Safety Measures for Blasting Practices

The following general safety measures are to be taken for blasting practices?

- A minimum radius of danger zone of about 500 m is to be

marked by special pointers to guard against the flying of projectiles, seismic effect and affect of air blast due to blasting. Before initiation of the explosives persons are to be posted in the strategic points to warn persons not to approach the danger zone. Safeguard flags may also be posted there.

- Safeguard flags must be posted at a minimum distance of 50 m from the blasting block when the explosive is delivered. All mining and haulage equipments must be taken out beyond this zone.
- Total blasting operation and supervision must be carried out by a team of qualified and experienced persons. Workers, supervisors and officers are to be imparted special training for this purpose.
- The operation of explosives should be carried out under the supervision of expert mining overman or foreman along with a highly experienced mining engineer. The officer and supervisor will check and note down the parameters of the depth of the holes, position and height of the charge and stemming, etc., into the blasting chart. If necessary they will change and reduce the amount of explosive into the holes. They will also check the sequencing of the blasting circuits and the continuity of the same.
- All the power transmission lines should be cut off and all the switches, etc. are to be removed from the blasting zone.
- The blasting operation should be carried out during the idle hours particularly at noon when density of the persons in the outside is very less.
- Based on the features/structures within the danger zone, decide upon the scheme of warning people within the said zone for ensuring their safety. The number of guards to be deployed for this purpose may be decided on the needs of the situation.
- Educate the people within danger zone about hazards from blasting and the need to take proper shelter at the time of blasting. Evolve a fool-proof method of warning them about impending blast as well as that of informing them about cessation of danger from blasting. But the difference between

'Danger Signal' and 'All Clear' signal should be made clear to all concerned.

- A suitable spot should be selected for firing shots. From this place the shotfirer should be able to communicate with all the guards deployed for ensuring that no person remains within the danger zone without due protection from projectiles. A walkie-talkie communication between shotfirer and guards would be disirable.
- The above drill is necessary in addition to :

  a) cereful planning of pattern and sequence of blast holes;

  b) correct charging and stemming of holes;

  c) proper sequence of blasting using short delay detonators, where necessary;

  d) muffling of shots where required; and

  e) an accurate record of each blast should be maintained to generate literature on the subject.
- When the officer or supervisors deputed for the blasting operation will satisfy regarding all safety measures taken accurately, then a clearance signal will be given to the blaster to blast the holes after taking proper shelter.
- Blaster himself should first take the proper shelter (shelter house) provided for that purpose and then he will blast the holes.
- Electric blasting operation should not be carried out during an electric storm, or in stormy or cloudy weather, etc.
- After blasting is over persons have to wait for a stipulated time, after which the blasting block should be examined thoroughly for any misfires.
- If any misfire is located then first it is to be death with and then ALL-CLEAR signal should be given.
- But when heavy blasting is involved in the opencast mines at a very great depth artificial ventilation is to be provided to dilute and disperse the noxious gases for several hours.
- To guard the structures located in the radius of the danger zone against the seismic action like ground vibration and

against the action of air blast, special precautionary measures like controlled blasting, etc are to be practised.

Question 148 : Discuss elaborately various problems and their solution after blasting in an overburden/ore bench.

Answer : Various problems and their solutions are as following :

Poor Fragmentation

Poor fragmentation will result formation of boulders and require secondary blasting. It is generally resulted when the face contains rock of different hardness in the same bench and also when the burden is more.

So proper designing of burden, spacing, bench height, increasing charge column with low density explosive, etc. will result better fragmentation and less fine production.

Ledge Formation

Ledges are formed generally if the drilling is not proper and secondary drilling and blasting is required to remove them. If inclined hole drilling is practised along with the direction of slope of the face and is blasted with optimum charging then this problem can be easily solved. However, when it is formed then additional holes should be drilled at an angle of 20^0 to 30^0 from the vertical along the direction of slope and is to be blasted.

Preparation of Drilling Site after Blasting

After blasting when the face is not cleaned it is very difficult to know the crest of the bench and hence without the help of survey department it is not easy to mark the points of the next drilling zone.

Toe Problem

When the rock is not homogeneous and the drilling and blasting is not proper then this problem may arise. To solve this problem adequate subgrade drilling, using high density and high VOD explosives at the bottom, proper firing sequence, proper charge density in the hole, bottom initiation with the help of Nonel of Raydet shock tube and inclined drilling in the front row of the drilling zone are worth to mention.

Back Break

Since the breaking of rock at the back side of the drilling block does not reach upto the floor level this will pose only problems for subsequent drilling, blasting and also to the excavator during loading.

This can be avoided by presplitting and controlled blasting by adjusting adequate charge density and maintaining an optimum delay interval between the last two rows of the holes.

Fly rock Problem

The factors on which flying of rock fragments depends are as following :

- large burden
- non-availability of free face
- irregular shape of the free face
- height of stemming column and stemming material in the blast hole. If the stemming height is 1.2 to 1.5 times the burden then flyrock nicely can be controlled.
- muffling over the blasting site and the muffling material.
- presence of water in the blast holes
- timing of delays of the detonator. A delay period of 10 to 15 ms per metre of burden may be optimum.

Muffling

Muffling of the blasting site is done in the following manner

- Lay steel wire expandable nets continually over the holes having the same delay number. Width of the wire net may be 1.5 m with the size of the aperture of 50mm x 50 mm.
- Place gunny bags filled with sand or drill cuttings, about 40 to 50 kgs in weight, over the wire nets at an interval of 3 to 4m.

Blast-Induced Vibration

The main factors which are responsible for blast induced vibration are as following

- Sub-grade drilling
- Charge depth, charge weight per delay detonator
- Explosive types and their properties
- Delay interval between the rows of the holes and between the holes in the same row.
- Height of stemming column and stemming material used
- By employing controlled blasting techniques
- Distance of the effected structure from the blast site

- Degree of muffling
- Powder factor
- Geology
- Physio-mechanical properties of the rock mass
- Irregular shape of the free face
- Due to large burden or blasting without a free face.

To reduce vibration the following measures may be adopted

- Use explosive with reduced weight strength and density
- Use multideck charge in a hole with optimised delay interval
- Presplit the last row of holes before final blasting for production.

Stray Current and Electrical Storms

During heavy storm, cloudy weather or rainy day problem of stray current and electrical storms may arise and during that time either blasting is to be stopped or blasting is to be done with non-electrical method.

Control of ground vibration near the locality and important place

Control of ground vibration is done in the following manner :

a) Presplitting of the drilling block following the air-deck method.

b) By controlled blasting method using lesser diameter drill hole, less spacing, burden, bench height, accomodating uniformly explosive into the blast hole, using optimum delay period, etc.

c) By using Hydraulic Hammers which produce no vibration during impacting on boulder or rock and can be attached to the boom of back-hoe shovel.

d) Ripping by the modern high power ripper.

Air Blast Problem

By controlled blasting this problem generally does not arise. However, during plaster shooting or pop shooting of the secondary breakage of the big boulders, this problem may arise. If Hydraulic Hammer which can be attached with the boom of back-hoe shovel is used for breaking the boulder, this problem can be nicely eliminated.

Question 149 : How does air blast occur? How can peak over pressure be determined? How can air blast be reduced?

Answer :

Air blast

When vibration waves travel through the air, it gives rise to an air blast. The airblast may occur due to following reasons :

(a) The movement of rock-mass generates the primary airblast. The velocity of face movement may be in the rate of 45 m/s to 55 m/s which compresses the air and the latter propagates as a wave away from the blast site. This pressure pulse is infrasonic (<20 Hz) and may be within the frequency range of residential structures which may amplify the pulse.

(b) Other pulses of air blast may be generated by a blast in the hole but have less intensity compare to the primary airblast. This air pressure pulse depends upon the ejection of material from the face due to inadequate burden, heaving of back rows of shot due to excess confinement of venting of material through the collar of a borehole due to inadequate stemming, etc. This air pressure pulses have higher frequency compare to the primary airblast pulse and may be accompanied by excessive dust, fly rock etc.

(c) Large low - frequency motion of the ground surface away from the blast site may generate an air pressure pulse.

These airblast generated by the above reasons may be modified by the wind pressure, temperature inversions and atmospheric conditions. Wind may add to an airblast pulse if it is moving in the same direction whereas the temperature inversion will reflect the airblast pulse.

The following equations hold good to determine the peak over pressure

(1) $$dB = 20 \log 10 \left(\frac{P}{P_O} \right)$$

(2) $$WdB1 = 164.4 - 24 \log \left(\frac{X}{W^{0.333}} \right)$$

(3) $$Poverp = 3300 \left(\frac{X}{W^{0.333}} \right)^{-1.2}$$

and

(4) Poverp = 0.38 PPV

Wdb1 = Over pressure decibel level (linear weighting)

(This equation is valid for the vibration induced over pressure)

Where

P = measured peak sound pressure

X = Distance of measuring point from the blasthole, m

W = Weight of explosive detonating per delay (Kg)

P_0 = reference pressure of 20 x 10^{-6} Pascal.

Poverp = Over pressure level (Pa)

PPV = peak particle velocity, m/s

This peak over pressure can be controlled by the charge weight of explosive/delay detonator and the distance from the blasthole. Air blast can be reduced in the following manner :

(a) introduction of air deck in each hole approximately half the amount of explosive in these holes relative to other holes

(b) reducing the blasthole diameter in the front row holes

(c) increasing the front row burden relative to the burden on all other rows.

Question 150 : Discuss the various blast monitoring techniques.

Answer :

The Blast Monitoring Techniques

The basic blast manitoring techniques are (i) Event monitoring and (ii) Performance monitoring.

Event Monitoring

Blast event monitoring technique uses a variety of sensors to detect the detonation or initiation of separate explosive charges. The detonation is sensed by various sensors e.g. vibration sensors, infra-red sensors, radio frequency detectors, pressure gauges, etc. The most common type of sensor is the vibration sensor.

The vibration sensors are attached to the rock near to the blast holes and detect the intense shock impulses produced by individual charges when they detonate. The commonly used sensores are geophones

(for velocity monitoring) or accelerometers (for acceleration monitoring). For higher shock resistance and frequency response accelerometers are the better option.

Limitation

- If all the charges use the same delay, the vibration based monitoring may not be able to detect the detonation of all the charges if the formers are all detonated simultaneously.
- Influenced by the properties of the rock to be blasted, especially the modulus of the rock surrounding the blasting.
- Where the delay interval between the successive charge detonation is less than the individual wavelet duration, interaction and enhancement of vibration will occur and the resulting vibration waveform can become very complex to segregate and recognize.

Performance Monitoring

It involves the collection of data that describe the performance of the explosive or the response of the rock to the explosives. These data can be used to quantify or compare explosive performance or blast pattern.

The parameters involve under heading are the measurement of fragment size distribution, blasthole velocity of detonation, burden movement velocities and displacement and levels of induced shock energy.

In the performance monitoring the following parameters are the most important :

Degradation : Degradation in the form of induced cracking will vary with hole diameter and the type of rock. With larger hole diameter cracks may extend at a distance exceeding 20 m.

Control measure : By careful designing blast to limit the pack particle velocity to 700 m/s in the volume of rock to be protected.

Displacement : Displacement of blocks produced either by delayed gas pressures or by vibration - induce shaking. It is associated with leakages of gas pressures along the open joints and poor shot design with large burdens. Vibratory displacement is associated with unstable blocks in rock slope and wherever ground motions produce permanent displacements, which are greater than the first-order asperity wavelength of rock joints.

Control measure : By making the sliding rocks stable and by providing adequate factor of safety to each block to prevent static failure.

Fly rocks : It is produced by explosive expulsion from the top of the blasthole.

Control measures : By adjusting the delay pattern, using decking, muffle blasting, control blasting, proper steming, proper blast design by adjusted spacing and burden, and by using shock tube detonators,

Soil densification : mass of loose and clean sand or soil can be densified by blast-induced vibration. Densification depends on the soil density, grain size, mineralogy, presence of cementing materials, etc.

Control measure : use slight cementing material with the soil.

Structural displacement : Associated with the vibratory response of facilities in or on the rock of soil mass surrounding the blast. The temporary peak displacement lasts less than one hundredth of a second and the structure returns to its original position afterwards. The effects are structural distortion, falling objects, formation of cracks, cosmetic craking oi wall coverings, human response micro-disturbances, etc. The structural displacement may be as follows -

- major - large cracks, shifting of foundation or bearing walls, weakening of superstructure, etc.
- minor - broken windows, loosen plaster, hairline cracks in masonary, etc.
- Cosmetic cracking - opening of old cracks, formation of new plaster cracks, dislodging of loose objects, etc.

Air Overpressure : Air pressure waves are generated by explosions. The higher-frequency portion of the pressure wave is audible and is the sound that accompanies a blast, the lower frequency portion is not audible but excites structures and in turn causes a secondary and audible sound within a structure.

Human response : Humans are very much sensitive to motion and noise that accompany blast induced ground and air-borne disturbances. Significant blast induced inaudible air overpressure and audible noise immediately follows the ground motion intensify human response. Tables 34 and 35 show the air over pressure limit and effect of air blast respectively.

Table 34

Air Overpressure Limits

USA	
Lower frequency limit of measuring system (Hz - 3dB)	**Maximum peak level (dB)**
0.1 or lower - flat response	134
2 or lower - flat response	133
6 or lower - flat response	129
C - weighted - flat response	105

Table 35

Effect of Air-Blast

Over Pressure		**Effect**
dB	**Kpa**	
128	0.05	Complains may be received
136	0.13	Allowable air-blast as per USBM
140	0.20	Dishes and loose windows rattle, some large plated glass windows may break
150	0.63	Some windows break
170	6.30	Most windows break
177	14.00	All windows break

Question 151 : Why are the following problems generated and how may be controlled?

a) Poor fragmentation,

(b) Ledge formation

(c) Toe problem

(d) Back break

(e) Strong current and electrical storms

(f) Air blast problem.

Answer : Consult the answer to Question 148.

Question 152 : On what factors flying of rock fragments depends? How can it be controlled?

Answer : Consult the answer to Question 148.

Question 153 : What are the main factors which are responsible for blast induced vibration? How may they be reduced?

Answer : Consult the answer to Question 148.

Question 154 : How may the ground vibration be controlled near the locality and important place?

Answer :

Some controlling measures of ground vibration :

The variables related to ground vibrations which can not be controlled are local geology, structure lithology, structural condition, the distance from the blast site, etc. Local geology has a strong influence in the transmission of vibrations involving both structure and composition. These factors must be taken into account during the design of the blast round and during selection of alternatives among the variables which can be controlled.

The controllable variables are the size of blast round, using of millisecond delay detonator/shock tube detonators, selection of delay, method of initiation of delay detonators, use of low detonation velocity explosives and low energy detonating cord, use of optimum stemming, covering of the detonating cord with soil, increasing delays between detonation, reducing spacing and burden, reducing the weight of explosives per delay, deck charging, etc.

Also consult the answer to Question 148.

Question 155 : Discuss briefly about BALSPA and SABREX Programme.

Answer : Latest development in the blasting technology involve E.T.I.'s (Explosives Technologies International) BALSPA and ICI's SABREX programme which are extensively used now a days to simulate rock blasting for the purpose of optimising driling and blasting cost. The BALSPA model depends upon hundreds of factors and can solve hundreds of types of problems after considering details of geological information, blast geometry, explosive types and characteristics, etc. This method saves considerable time, effort, costs, etc. It can study effectively the effect of V.O.D.; angled hole delay hook up, collar, hole size, bench height, spacing, decking, subgrade drilling, water, distribution of explosive, choose the type of explosive for a given rock, delays necessary, study new blasting techniques, etc.

SABREX (Scientific Approach to Breaking Rock with Explosives), the micro computer model developed by ICI can rapidly simulate blasts and combines computer graphics with the analytic approach of the interactions of explosives with rock. Three essential elements of the SABREX include blast geometry, detonation peoperties of explosives and strength and elastic properties of rock. To evaluate the blast design inputs of Rock Properties for the SABREX is density, unconfined dynamic tensile strength and static compressive strength, young modulus, poissons ratios, etc. are very important. Blast Geometry includes drilling pattern, inclination, bench height, burden, spacing, sub-grade initiation pattern, delay, etc. whereas explosive parameter includes density, detonation velocity, detonation pressure, explosion pressure, shock energy, gas energy, etc. The output of SABREX will give the profile of muckpile fragmentation size, data of fly rock, back break, cost of drilling and blasting, etc. which will save considerable time, efforts and costs for designing most effective blasting system.

Question 156 : How do misfires occur in the surface mining? How is it to be dealt with?

Answer : Misfires occur in the surface mining due to the following reason :

a) In firing with electric detonator

- Poor priming
- Faulty detonators
- Damaged explosives
- High resistance in the detonators
- Blasting more number of shots beyond the capacity of the exploder
- Loose connection
- Insufficient firing current due to leakage or short circuiting
- Damage of explosive in the presence of water in the shothole
- Geological disturbances
- Fault in the blasting cable
- Incorrect selection of delays and other initiation system.

b) In firing with safety fuse

- Due to improper crimping moisture enters into the fused cap

- Due to damaged, kinked, broken or wet fuses.

Dealing with misfires

As a safety rule no person will be allowed to enter the blasting site before the expiry of 5 minutes in case of electric shot firing and 30 minutes in case of safety fuse firing. After the stipulated time is over the blaster/blasting foreman/blasting engineer shall go to the blasting site and search for any misfire. If any misfire is found then it must be dealt very carefully since it is a very hazardous operation. So it is adviseable that the dealing of misfire must be conducted under the supervision and direction of the shift under manager. The under manager shall be assisted by the blasting engineer, overman/foreman and blaster. The details marked in the drilling and blasting plan and actual cause of misfire shall be thoroughly discussed to find out the best solution of the problem. As a safety guard the HEMM to be involved for dealing the misfired shot hole (s) must be the minimum. They shall be kept at the safest distance and all windows of the shovel/front end loader shall be well shielded by attaching extra strong shielding net/cage. The whole operation must be carried out in the day light. The misfired site should be properly fenced and demarcated with red flags.

The dealing of misfires in the surface mining system are as follows :

- First of all determine the number of misfired shotholes, findout amount of explosives in each shothole and the total amount of unfired explosives.
- In case of misfire due to faulty connection, cutout of detonating cord then do the following :
  1) First of all check out that the existing burden is not reduced to a level which may cause fly rock if refired.
  2) In case of an electric shot firing the leg wires of the underground blastholes shall be reconnected with a connecting wires and then other end of the connecting wires shall be connected to a blasting machine and shall be fired after taking proper shelter by the shotfirer.
  3) In case of misfired shothole is due to the detonating cord then tape an electric detonator at the end of detonating cord. Connect the connecting wires to the leg wires of the detonator and attach the other of the connecting wires to

the blasting machine. Blast the shotholes after taking proper shelter.

- In case of the misfire shotholes are due to other reasons then the following procedures may be followed :

  1) Remove the stemming materials with an air-compressor by placing the blowpipe in the shotholes after duly earthing the nozzle and blow out the stemming materials. Take a suitable primer, make it up, lower it down into the blasthole over the top of the explosive, stem the blasthole with proper stemming material and fire the blasthole after duly taking proper shelter.

  2) In case of explosives has to be removed from the misfired shotholes then either of the following ways may be followed :

     (a) After removing the stemming materials with the help of same compressed air system HANFO, bulk explosives, ANFO etc. can be removed.

     (b) Cartridge explosives may be removed by pulling them out with leg wires, or by hooking up with steel clip attached with a rope or by wedging a pointed stick into the cartridge and afterward pulling it out.

  (c) Misfired shotholes may be uncovered by shovel or dragline. Once the explosive column is uncovered the dragline or shovel is to be stopped and explosives shall be blown out by using the air compressor.

  3) It ANFO is used as column charge then it may be damped by pouring water into the blasthole after removing the stemming material.

  4) If cleaning of shotholes by the flushing out by the compressed air or water jet is not possible then a second reliving shothole parallel and equal depth of misfired shothole, is to be drilled away from the each misfired shothole. They are to be charged and blasted afterward. The second shothole generally spaced by a distance of 2.5 m from each misfired hole.

After blasting is over the site shall be well examined by the Under

manager/Blasting engineer/Blasting foreman. Appropriate action shall be taken to collect the debries by the shovel/front end loader and shall be loaded over dumper/tripping trucks and the debries shall be removed from the site. This search operation shall be continued till any portion of the misfrired charge is completely removed. All such operations must be carried out after taking all due safety precautions.

For more details, consult answer to Questions 175 and 176.

Question 157 : Enumerate some salient points regarding the safe practices during blasting in opencast mines.

Answer :

Be Safe

Don't—

- expose explosives to excessive heat,
- keep explosives in the lamp room or other unauthorized places,
- shoot a bullet on to them,
- carry/use damaged/deteriorated explosives, detonating cords, detonators, exploder, blasting cables, and other accessories,
- fight a fire while carrying explosives,
- smoke and also don't allow anybody to smoke near to an explosives,
- allow any source of ignition within 50 m of a blasting area or 25 m of a magazine, reuse packaging for explosives,
- allow growing of any grass, herbs etc. within 15 m surrounding the area of the magazine,
- remove explosive materials from it's packaging unless the former is meant to be used in that manner,
- inhale explosive material, dust or vapour, undergo continuous skin contact with the explosives,
- after handling the explosives take food without washing hands,
- use any explosives and their accessories unless techniques of uses, rules, regulation, etc. are thoroughly known and highly trained in this regard,
- keep magazine without lock and key except during issuing of explosives,

- handle explosives during an electrical storm in the opencast mines,
- leave any partially used explosive package unopened,
- carry detonators carelessly and not under lock and key,
- fire a blast until all is clear, sufficient warning is given by siren/microphone/other audible devices, watchmen are posted in all the vaulnerable points, red flags are properly shown and thereafter it must be assessed that everything is clear and O.K. in all respects,
- fire a blast without taking into a proper shelter at a sufficient distance from the blast site. In case of opencast workings shelter must be of an approved type blasting shelter,
- drill a hole into the previous blasted socket,
- pick out explosive charge after a misfire but deal it as per the rules, regulation and standard procedure,
- overload the holes,
- tamp the primer
- tamp the main explosive with tools made of sparking materials,
- use detonators from different manufacturers in the same circuit,
- while dealing with the explosives put off all the electrical equipment near the vicinity of the blast-holes,
- keep the leg-wires of the detonators unshunted,
- try to fire without testing the resistance, continuity, etc. of the blasting circuit with the approve type apparatus.Check stray current with multimeter, blasting circuit's continuity and resistance by galvanometer, ohmmeter or multimeter,
- over load the exploder,
- allow bare wires touching the ground,
- use safety fuse less than 1 m length and is kinked, bent or damaged,
- use other lighter except hot wire fuse lighter to light the fuse,
- store explosives and detonators in the separate magazines,
- store ANFO and other explosives in the same place,

- store the cases or boxes of explosive materials at the ends or sides of the magazine and the boxes height shall not exceed 2 m,
- allow unauthorized entry, any spillage or leakage in the magazine,
- keep legwires and leadlines unshunted until just about to connect,
- use any circuit which is not made properly,
- load boreholes near the electric power line,
- keep any part of the firing circuits closer to any conductor such as bare wire, rails, pipes, wet ground, sources of any stray current, etc.
- fire the charge during dust, snow or electrical storm or when an electrical strom is approaching,
- keep on connecting the leadline with the exploder after blasting is over,
- coil the blasting cable without shunting the ends,
- return to the blasting site and near to the misfired holes before 15 minutes in case of electric firing and 30 minutes in case of firing by the safety fuse,
- return to the blasting site while explosives in the holes are burning. All the persons must be kept away for atleast one hour,
- return to the site till the loose materials in the high walls are cleared off,
- move into the face before clearing off any smoke, dust, fumes, and undetonated explosives by the competent person (s),
- try to destroy unusable explosives without consulting the manufacturer,
- reenter into the site or unblock the vulnerable points by the guards without listening the sound of "All-Clear" signals.

Questien 158 : Write short notes on latest technology of quarry face and borehole surveying.

Answer : The latest instrument which is used in quarry face and bore hole surveying system is named as 'Quarryman^{TM} Face Master System'

which is an integrated surveying system comprising laser electronic and computer techniques. The instrument facilitates the blasting engineer and blasting foreman or overman to visualise completely the rock face to determine accurately the face profile, to plan the pattern of blast hole with optimum burden, to survey the total length of the borehole, to calculate the deviation of bore hole, ete. The quarryman™ face master system comprises the following :

- a quarryman™ laser rock profiling system
- an IBM PC computer system
- a quarryman™ FACE-3D software package
- a boretrak™ borehole survey system

The operators point the laser by the heavy duty laser theodolite at the face from any point within 500 m radius for measuring directly the bearings, distances, horizontal and vertical angles. Laser scans are made directly across the rock face until sufficient readings have been obtained for accurate defining the features of the face, quarry floor and two face line reference markers. Measurements are automatically recorded in the Quarryman™ electronic note book and the survey data are downloaded into the PC computer system for processing on the site with the help of a high speed 286 Laptop PC system. This will reduce the operational delays. However, if a high speed 386 desk top PC is used then high resolution colour graphics allows visualisation of the rock face geometry directly. The editing facilities are also provided in the PC compatible computer system. The Quarryman™ Face 3D software system processes and integrates the raw data from the Quarryman™ system, manipulate the data for optimistic face design, produces graphic display either in two or three dimensions and also provides concise survey documentation. The software package removes the observation errors of specific face profiles, can be interpreted for designing new profiles which eliminate the need to resurvey the profiles, correlates accurate relationship between the inter-profile and inter-borehole relationship, can calculate the actual production capacity, assesses hole loading, the blast ratio, etc. The blasting parameters can nicely be planned and computed for better blasting performance. The borehole deviation can be surveyed by the MDL's Boretrak™ system (dia. of the sensor probe is 38mm) in all the dry and wet bore holes having diameter 45mm or more down to a depth of 100m. The Boretrak™ consists mainly of a two axes measuring head

constructed from stainless steel and aluminium (1m length), a set of lightweight fibre glass alignment rods (each of 1 to 2m length), a battery pack, an electronic note book and a PC computer software system. The electronic gravity sensor mounted in the measuring head is lowered down into the hole by connecting it with the lightweight rods and the sensor determines the inclination, compensates for temperature and voltage fluctuations, stores calibration data, formats and controls data output. The fibre rods flex but do not twist enabling each reading to be related to the face orientation at the surface. Sensor depths are indicated by counting the number of rods in the hole. Depth is logged at an interval of 1.2 or 4m by pressing the electronic note book **enter button** until the bottom of the hole is reached. On completion of the survey, data from measuring head and electronic notebook are downloaded to PC Computer system when depth and integration takes place. With the help of standard stand alone Boretrak^{TM} PC software system section displays and plots are provided to give the operator visual as well as alpha numeric data about the hole position. Hole pitch and roll angles are automatically recorded at three second intervals. The Quarryman^{TM} face master system surveys entire rockface in minutes with safety (it takes about 10 minutes to survey a bench having width 15m and length 50m), helps designing optimum borehole arrangements with the help of concise and accurate graphics and alphanumeric values which will reduce drilling time, explosive cost, man hour to survey, insurance premium etc., increase safety by optimum designing the burden which will reduce fly rock, noise, air blast, provides better safety to survey personnel since they need not to go near to the face, yield more volume of rocks or mineral with better fragmentation which will reduce the wear and tear of the mining mechinery.

The Boretrak^{TM} system facilitates to adjust accurately the borehole charging parameters, planning of good blast design and provides guidance to the drillers regarding the deviation of the blast holes.

●

Chapter 3

Blasting Practices in Underground Coal Mines

Question 159 : What are the characteristics which govern the selection of an explosive in underground coal mine? Categorise the permitted explosives with examples. What is meant by permitted explosives? What types of permitted explosives are used in coal seams of gassiness degree I, II, and III? Give some examples of permitted explosives manufactured by IEL enumerating their various properties and uses.

Answer : Since the last few decades great advances have been made in the technique of rock blasting as a result of greater understanding of the mechanism of rock breakage and the development of sophisticated sequential firing techniques. A comprehensive range of explosives and accessories has been developed to suit the modern trend in blasting. The characteristics like strength, velocity of detonation, density, water resistance, sensitivity, fume characteristics, detonation pressure etc. are the prime factors which govern the selection of an explosive for a particular purpose. Permitted explosives are developed in great extent as a safe guard against the fire damp and coal dust explosion in underground coal mines which are categorized as following :

Group P1 Ordinary permitted explosives. These are mainly used in drifting and pulsed infusion shotfiring. Example : Ajax - G, Viking - G, Godyne.

Group P2 Sheathed permitted explosives. These may be used for instantaneous or simultaneous firing in coal or ripping with appropriate safeguards.

Group P3 Equivalent sheathed explosives (EQS). These may be used where sheathed explosives are approved. Example : Unisax - 'G', Permadyne, Unipruf -'G'.

Group P4 These are semi ion exchange explosives and are suitable for delay firing in ripping and cross-measure drifts. Examples 'Carribel' and 'Carrirax' etc.

Group P5 These are suitable for solid blasting. Example : Soligex, Pentadyne etc.

Permitted explosive means an explosive manufactured by such firm and of such type as the Chief Inspector may from time to time specify by notification in the official hazette. In India all underground coal mines have been classified in terms of potential gas hazard and are required to use permitted explosives only. In coal seam of gassiness degree I, P1/P3/ P5 permitted explosives are used. Table 36 shows the various types of permitted explosives which are manufactured by IEL.

Table 36

PERMITTED EXPLOSIVES (Manufactured by IEL)

		PROPERTIES				
TYPE	TRADE NAME	Relative Weight Strength (% BG)	Relative Bulk Strength (%BG)	Density g/cc	Velocity of Detonation (metres/sec)	MAIN USE
NITRO-GLYCERINE GELATINES	Ajax G	54	55	1.55	2400	Drifting, ripping, blasting hard coal in gassy coal seams of first degree.
	Unisax G	41	44	1.60	2400	Drifting, ripping, blasting coal in all gassy coal seams.
NITRO-GLYCERINE SEMI-GELATINE	Soligex	39	33	1.30	2000	Delay blasting of coal off-the-solid and conventional blasting of coal in all gassy coal seams.
NITRO-GLYCERINE POWDERS	Viking G	60	40	1.00	2000	Drifting, ripping, blasting coal in all gassy coal seams.
	Unipruf G	46	29	0.97	1800	Drifting, ripping, blasting coal in all gassy coal seams.

Question 159 (A) : Discuss briefly regarding the permitted emulsion explosives developed for the use in the underground gassy coal mines.

Answer :

Recently ICI Explosives developed small diameter permitted emulsion explosives for using in the underground gassy coal mines. These type of explosives offers the following advantages—

- Because of rigid cartridges they are easy to handle, prime and load in the shot holes.

- Superior energy efficiency and performance through maximum realisation of potential chemical energy into useful work.
- Superior reaction efficiency resulting in minimum post detonating fumes.
- Unmatched water resistance for reliable results in wet conditions.
- Ultimate in safety through reduced sensitivity to impact, abraison or heat.

The following (Table 36 A) is the brief specifications for the POWERGEL P101, POWERGEL P301, and POWERGEL P501 permitted emulsion explosives manufactured by the ICI Explosives—

Table 36 (A)

TYPE	Nominal density (g/ml)	VOD (m/s)	Relative effective energy	Relative bulk Strength (% ANFO)	MAIN USE (% ANFO)
POWERGEL-P101	1.15 ± 0.05	3500 ± 200	99	134	Coal seam of Ist degree gassiness. Blasting with undercut/midcut/overcut with the use of approved non-incendive instantaneous electric detonator only.
POWERGEL-P301	1.15 ± 0.05	3500 ± 200	98	133	Coal seam of Ist and 2nd degree gassiness. Blasting with undercut/midcut/overcut with the use of approved non-incendive instantaneous electric detonator only.
POWERGEL-P501	1.15 ± 0.05	3000 ± 200	56	79	For blasting coal off the solid or blasting with undercut/midcut/overcut with the use of approved non-incendive instantaneous electric detonator only.

Question 160 : With the help of neat sketches show the preparation of a primer cartridge. With the help of sketches explain 'Direct Initiation' and 'Inverse Initiation'.

Answer : After preparing a primer cartridge as shown in the Fig. 62(a),

(b), (c) and (d), first of all insert a cartridge of stemming material in the shothole which will be followed by the ordinary explosive cartridge. The primer cartridge will be pushed at the last as shown in the Fig. 62 (i). It is called the 'direct initiation', whereas, when the 'primer cartridge' is at the back of the charge and the orientation of the detonator points towards the front of the hole as shown in the Fig. 62(ii), is called the 'inverse initiation'.

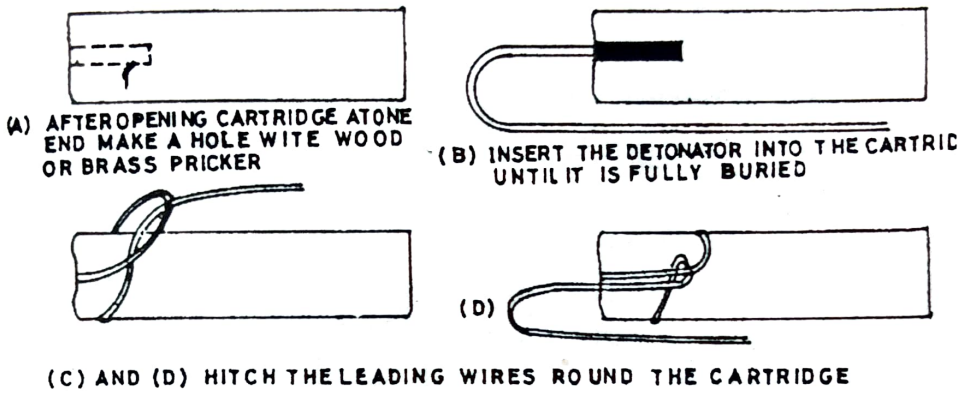


Fig. 62 Preparation of Primer Cartridge

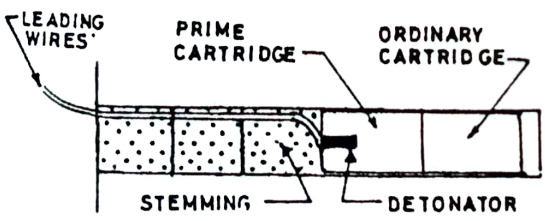


Fig. 62 (i) Direct Initiation

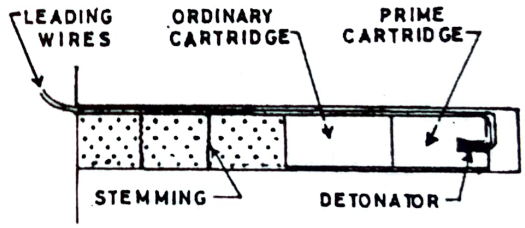


Fig. 62 (ii) Inverse Initiation

Question 161 : Discuss the factors which govern the drilling and blasting pattern in coal. Under which condition undercut, middle cut and overcut is given? Discuss charge per hole, number of rows of holes during blasting coal faces. Show typical blasting patterns in the following cases :

1. Development heading having an undercut
2. Development heading having a middle cut
3. Development heading having a middle cut and hard stone band.
4. In a longwall face having under cut.

Answer : The factors which influence the drilling and blasting pattern in coal are as following :

- Nature of coal

The presence of cleats or cleavages and hardness of coal determine its breakability.

- Nature of roof and floor

  When floor is soft under cuttings is very difficult with CCM where as when roof is weak and charge is not controlled there may be chances of excessive shattering.

- Nature of coal seams

  The thickness of coal seam; the presence of dirt and stone band, hard coal bands etc. affect the performance of explosives and hence are critical in designing the blasting pattern.

- Gradient of coal seams

  If the gradient of coal seams are more than 15^0 under cutting is very difficult and solid blasting may be the only practicable alternative.

- Structural disturbances in coal seams

  Folds, faults, dykes and sills in coal seams also affect the normal working pattern.

In a coal drivage when an undercut is given by a CCM to provide an additional free face the length of the cut should be at least 15cm larger than the length of the hole. However in dip galleries where the accumulation of water poses problem "middle cut" is often preferable. When a soft shale or stone band is present in coal; cleaner coal may be obtained by cutting into such a band whereas when a weak roof is to be encountered an 'overcut' is preferable to minimise the shock of blasting. The depth of cut normally varies between 1.2 to 1.8 m although a depth of cut upto 2.7 m is possible. The burden on any hole should always be less than its depth, the best results are obtained when the burden is about 2/3rd of the depth of the shothole. The holes at the top row (in case of height of the gallery is more than 1.5m) should be directed towards the stone band to break it smoothly with more force and the holes should penetrate the cleavaged plane at right angles. The maximum charge per hole is limited to 1 kg in the case of P3 and P5 explosives and 0.79 kg in the case of P1 explosives. In multishot firing the detonators must be connected in series and fired simultaneously. This eliminates chances of a 'cut off'.

Upto a seam thickness of 1.2 m a single row of holes is sufficient whereas for thicker seams more rows may be necessary. The depth of holes normally lies between 1.2 to 2 m whereas the spacing and burden on the holes lying between 1 to 1.2 m. Fig 63, 64, 65 shows typical blasting patterns in the drivage in coal whereas Fig. 66 shows the typical blasting pattern in a longwall face.

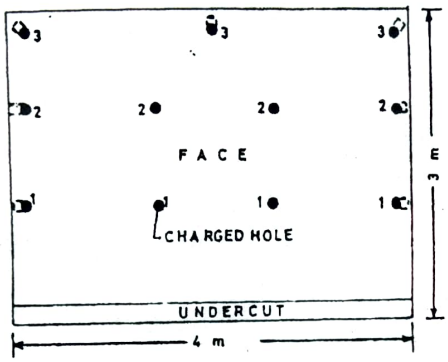


Fig. 63 Typical Drilling Round of Size 4m wide x 3m High (under cut)

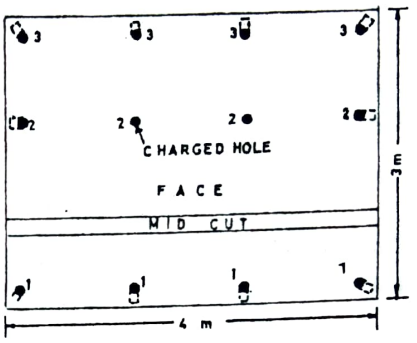


Fig. 64 Typical Drilling Round of Size 4m wide x 3m High (Mid cut)

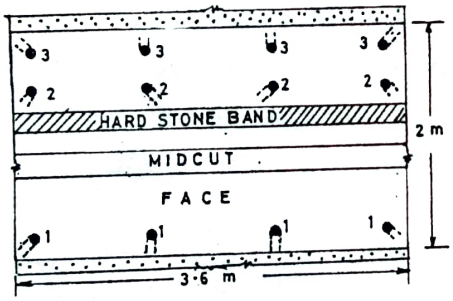


Fig. 65 Typical Drilling Round of Size 3.6m wide x 2m High (Mid-cut) with stone band

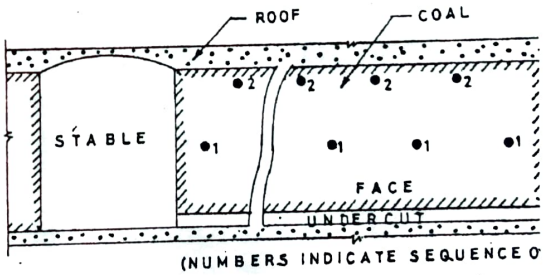


Fig. 66 Typical Blasting pattern on longwall face

Question 162 : What is blasting off the solid? What cut patterns are generally used in this case? Show a typical blasting pattern in a longwall face. During adopting the solid blasting technique in a coal mine what salient technical points are to be considered in depth?

Blasting off the solid

Both in the longwall (Fig. 67) and in the development faces coal can be blasted without giving an undercut by a coal cutting machine and the completed blasting operation is carried out by the use of only P_5 type explosive. The blasting operation is termed as blasting off the solid. In solid blasting the initial cut is created by using either wedge cut, fan cut or pyramid cut pattern, towards which the rest of the shots are fired. The technique of blasting off the solid is simpler, more economical and less

hazardous than the conventional method of breaking coal with machine

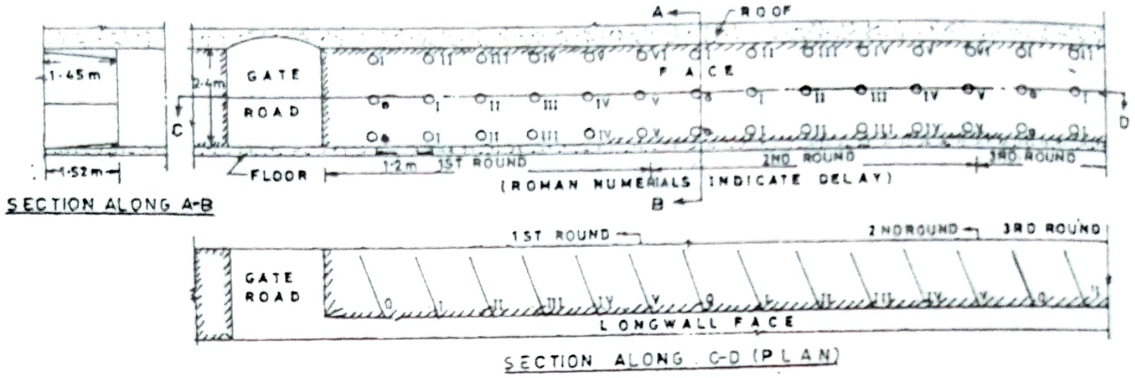


Fig. 67 A typical layout of shotholes in a longwall face (Blasting coal off the solid)

cutting and blasting. The following are the salient technical points which are to be followed to adopt solid blasting in a coal mine :

- No explosive other than P5 explosive and no detonator other than 'Non-incendive' short (millisecond) delay detonators shall be used for this purpose.
- In devising the pattern
  a) The estimated period or delay between the first and last shots of the round calculated by reference to the marking on the detonators shall not exceed 0.15 second (150 milli-seconds).
  b) The estimated period of delay between any two consecutive delay shots shall not exceed 0.06 seconds (60 milli-second).
  c) The minimum distance between two adjacent shot holes of different delays shall not be less than 0.6 metres, between the portions of shot-holes containing the explosive charge.
- At least 284 cubic metres of air per minute shall be conducted upto every face where solid blasting is being done.
- Tests for inflammable gas shall be made at and within a radius of 20 metres of the place of firing with an approved type of methanometer by a person holding a manager's, or overman's certificate and no shot shall be fired if 0.5% or more inflammable gas is found at any place in the said zone.
- The shot-firer
  a) shall not fire more than 25 shots in one round.

b) shall strictly adhere to the pattern of shot-holes and charge per hole etc. laid down by the manager and

c) shall carry out all statutory inspections and precautions for inflammable gas etc.

- The roof, floor and sides of all workings within 20 metres of the place of firing shall be thoroughly drenched with water.

a) The exploder shall be of an approved type and shall be maintained in perfect working condition.

b) The shotfirer shall be warned to take due precautions for the high voltage output of the exploder while handling the same.

c) The cable for shot firing shall be of an approved type and adequate for the maximum voltage output of the exploder.

d) All connections between the shotfiring cable and the detonator leads as well as between the consecutive leading wires shall be adequately insulated.

Question 163 : How is the free face created in blasting coal off-the solid? What patterns of cuts are most suitable in solid blasting? With the help of neat sketches discuss the following

a) The wedge cut pattern

b) The fan cut pattern

Why is solid blasting still practised?

Show a typical solid blasting pattern in a development face having a stab hole.

Answer : The general practice of mining coal by machine-cutting and blasting involves creating a second 'free face' for better utilization of explosives energy by providing a suitable cut using coal-cutting machine. This is followed by blasting with Pl/P3 explosives (depending upon the degree of gassiness) and instantaneous copper electric detonators. In blasting coal off-the-solid, the coal cutting machine is eliminated as the second 'free-face' is provided by creating an initial opening, just as in tunnelling operations, by firing a few holes and then blasting the remaining holes into this opening. Both these operations are combined in one round by the use of delay detonators. The holes for initial opening can be arranged in any of the following two patterns which have been

found to be most suitable :

a) The Wedge Cut Pattern

b) The Fan Cut Pattern

a) Wedge Cut Pattern

In a 'wedge cut', two or more holes are drilled so as to converge at the rear by not less than 0.3 m. This arrangement provides a concentration of charge at the back, and when these holes are fired with the same delay number a wedge-shaped opening is formed. With this pattern, the advance is limited and for deep pulls burden should be reduced by providing a steeply inclined hole at the centre, known as 'stab hole' or by giving another shallow, 'wedge cut' known as 'baby cut'. These relieving holes should be fired prior to the main 'wedge cut' pattern.

A typical example of a pattern based on 'wedge cut', which is being widely used in a number of mines in India, is given in Fig. 68.

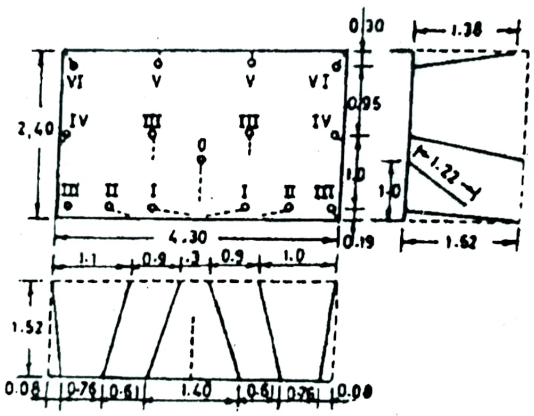


Fig. 68 Wedge Cut

This pattern is suitable for soft and medium-hard coal but has its limitations in very hard coal and in seams which gave highly pronounced cleavage planes.

The average performance of the pattern based on 'wedge cut' is summarised below :

Size of gallery	:	4.3m x 2.4m
No. of holes	:	15
Total charge	:	6-7 kg
Blasting ratio	:	Avg. 2.6 t/kg
Yield per detonator	:	1.1 to 1.2 t
Pull	:	1.38 - 1.52 m

b) Fan Cut Pattern

In fan cut pattern, the initial opening is created by firing a hole drilled at an acute angle to the face. The inclination generally varies from 30^0 to 60^0. The inclination of subsequent holes is gradually reduced. In most of the cases, the burden of the initial holes is kept below 1.2m. This pattern is suitable for medium hard to hard coal and in seams having pronounced cleavage planes. Besides, a comparatively better advane per round can be achieved with this pattern. Fig. 69 illustrates a pattern based on fan cut.

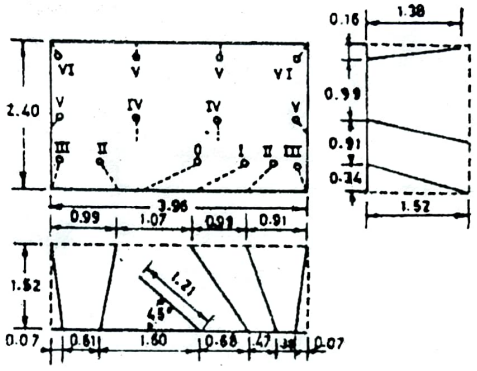


Fig. 69 Fan Cut

The average performance of the pattern given in Fig. 69 is as follows :

No of holes	:	14
Charge : Bottom Row	:	500 to 550 g
		Rest of the holes : 350 to 400 g
Blasting ratio	:	Avg. 2.7 t/kg
Yield per detonator	:	1.2 to 1.4 t
Pull	:	1.38 to 1.52 m

Although solid blasting requires about 30% more drilling than conventional machine-cutting and blasting, the versatility of the BOS technique has presented a non-cyclic system which has been tested and successfully employed over the past decade. It can be employed even in conditions where the floor is uneven or the seams are disturbed geologically and where coal-cutting machines are difficult to operate. The horizon of the face or the gradient can easily be maintained and floor blasting which cannot be avoided with the use of coal cutters can be eliminated. The higher rate of production with solid blasting also provided for a better face OMS. However, the efficacy of the technique

very much depends on the creation of initial opening which in turn depends on the accuracy of drilling of the 'cut holes'. A drilling and charging pattern for blasting of the solid is shown in Fig. 70. The details of blasting parameters are as following :

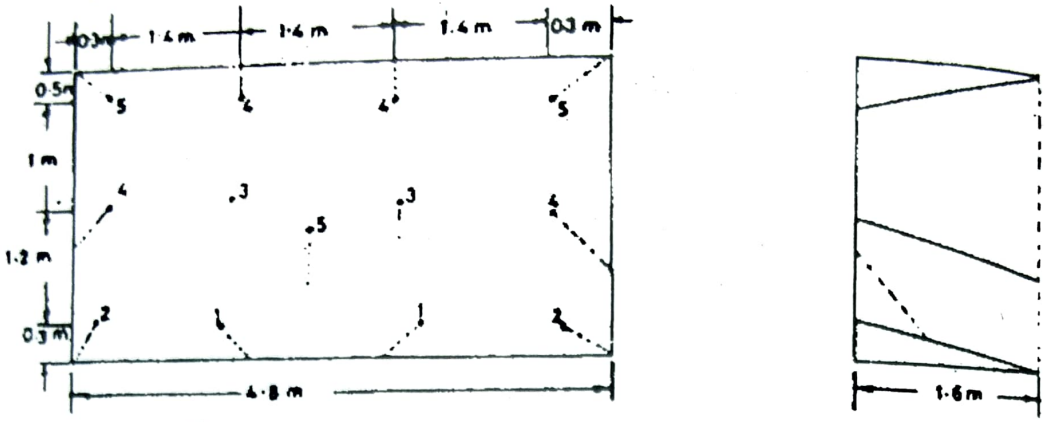


Fig. 70 Drilling and charging pattern (1, 2, 3.....indicate delay number) for blasting of the solid

No. of holes : 12, No. of stab holes : 1

Charge per hole

Delay No.	Explosive in gms
9, 1, 3	400 gms
2, 4, 5	600 gms

Total charge : 6.8 kg Explosive used : Soligex

Deronator - Non incendive copper delay

Maximum charge/delay - 2.4 kg, Av. coal obtained 16 - 18 tonnes

Powder factor : 2.35-2.6 tonne/kg, Average pull obtained : 1-1.1 m.

Question 164 : With the help of neat sketches discuss elaborately about the blasting of ring holes in blasting gallery method.

Answer :

Blasting of Ring Holes in Blasting Gallery Method

The pattern of drill holes and charging with spacers are shown in the Fig. 71 and 72.

The holes are drilled by means of electrohydraulic operated jumbo drill. The drill rods used are in segments and jointed by means of male to female joint. The speed of drilling is roughly 1 m/minute and water used to flush at a pressure of 200 kg/cm^2 (approx.). The number of drilled holes in a ring varies in between 40 to 45 nos. Length of the holes varies

from 3 m to 10.8 m (approx.) to blast a total coal seam height of 11 m including the height of the gallery (3 m). Special type of P3 explosive

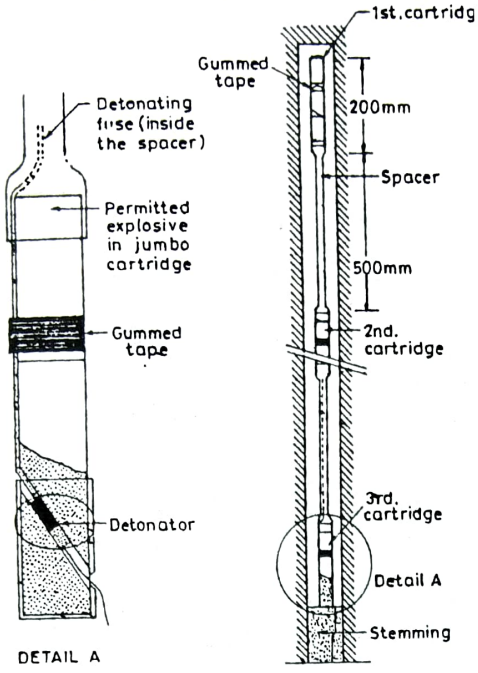
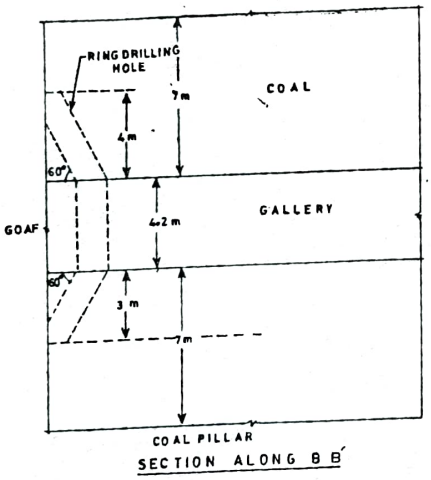
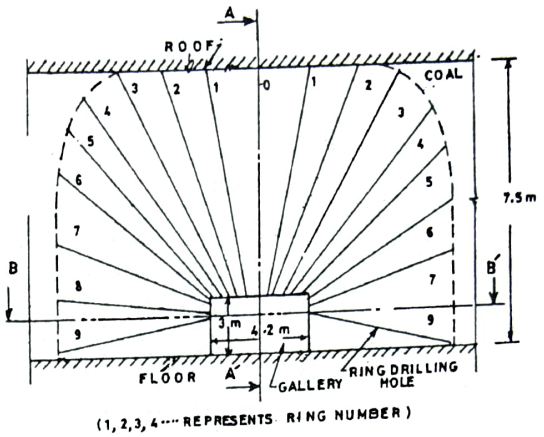


Fig. 71 Charging of a long hole in the Blasting Gallery Method





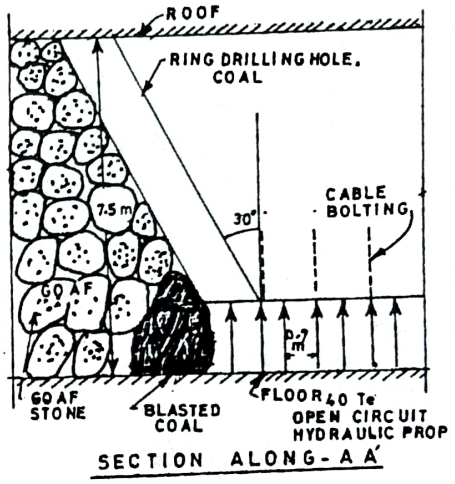


Fig. 72 Drilling Pattern in the Blasting Gallery Method

class 'Uni- ring' and special detonating fuse 'Ring-cord' are used for this purpose. The charge of explosives (cartridge are 32 mm dia and weighing 200 gms each) is inserted into the holes by means of plastic spacers. Explosive charge should not be more than 3 kg in longer holes. The distance between the two rings are 1.5 m. The holes of the rings are inclined at an angle of 30^0 to 50^0 (approx.) from the vertical towards the goaf. The details of the hole length and explosive used is given in the following table-37.

Table 37

Hole No. (Their respective length, m)	Explosive per hole (kg)
1(3.5) 2(6.35), 3(6.3), 4(6.35), 5(6.6)	1
6(6.85), 7(7.3), 8(8.0), 9(8.0)	2.5
10(8.4), 11(10.8), 12(10.8), 13(10.37), 14(10.2)	3.0
15(9.83), 16(9.8), 17(9.46), 18(9.6), 19(9.22), 20(9.9), 21(9.10), 22(9.0), 23(9.1), 24(9.9), 25(9.22), 25(9.5), 27(9.46), 28(9.8), 29(9.83)	2.5
30(10.2), 31(10.31), 32(10.8), 33(10.8), 34(8.4)	3.0
35(8.0), 36(8.0), 37(7.3), 38(6.85)	2.5
39(6.5), 40(6.35), 41(6.3), 42(6.35), 43(3.5)	1
Total	97.5 kg.

After charging is over all the shotholes should be plugged with the mixture of plastic clay and sand.

Question 165 : Give an example of blasting technique in a Blasting Gallery method of working.

Answer :

Example

Blasting Gallery Method in one of the SCCL Mines

Diameter of the ring holes : 43 mm

Average number of holes drilled per ring : 33

The distance between the two rings : 1.5 m

Learing of each ring towards the goaf : 30^0 to 40^0 from the vertical.

Drilling time + positioning time from one hole to another + setting time of Jumbo drill : 265 + 50 + 45 = 360 minutes

Total length of ring holes : 265 m

Explosive used : Uniring (P_3 Explosives)

Total charge in a ring : 83.4 Kg

Total ring cord : 250 m (ring cord is made with 4.5 gm of PETN/m length)

Total number of spacers used : 279
Number of adhesive tape used : 1
Production per blast : 306 tonne
Yield/hole : 9.3 tonne
Powder factor : 3.67 tonne/Kg of explosives.

The charge is prepared at the site with the explosives cartriges, spacers, detonating cord, adhesive tapes, etc. according to the length of each hole. Before loading holes, the methane content will be checked in the room, including the goaf. Then holes are loaded by a succession of cartridges and spacers linked together with detonating cord fitted along the entire length, from the first cartridge to the detonator which is inserted in the last cartridge. Charging is done with direct initiation. The maximum permitted charge per hole is 2 Kg. After charging and stemming of all the holes is completed, the circuit is connected in series. Before blasting, recovery of last installed roof support has to be made. The total duration of loading, stemming and blasting of a complete ring shall be about 3 hours and will require 4 shotfirer-helpers. Table : 38 depicts the details regarding the blasting.

Table 38

Hole Type	Number of hole	Length (m)	Number of Cartridges per hole	Number of spacers per hole	Explosive per hole (kg)
1	1	9.00	13	11	2.60
2	2	9.30	13	11	2.60
3	2	9.50	13	11	2.60
4	2	9.80	14	12	2.80
5	2	10.20	15	13	3.00
6	2	10.80	15	13	3.00
7	2	10.80	15	13	3.00
8	2	8.40	15	13	3.00
9	2	8.00	14	7	2.80
10	2	8.00	14	7	2.80
11	2	7.30	12	6	2.40
12	2	6.85	12	6	2.40

Table - 38 (*Continued*)

Hole Type	Number of hole	Length (m)	Number of Cartridges per hole	Number of spacers per hole	Explosive per hole (kg)
13	2	6.50	11	5	2.20
14	2	6.35	11	5	2.20
15	2	6.30	11	5	2.20
16	2	6.35	11	5	2.20
17	2	3.50	6	2	1.20
Total	33	265.00	417	279	83.40

Question 166 : What safety measures are to be considered during blasting practices in Blasting Gallery Method of extraction of pillars?

Answer : The following are the basic points which are to be considered during ring drilling and blasting in blasting gallery method of extraction of pillars.

1(a)Advice of CdFI France and IEL Ltd shall be obtained in evolving the system of long hole blasting and in training of personnel.

(b) No person shall be employed on any work unless that person has received the necessary instructions and training so as to be able to do the work competently and safely.

(c) The manager shall prepare a scheme for training that would cover all operations involved in the extraction by long hole blasting method.

2(a)Ventilation shall be adequate to bring down the concentration of NO_2 and carbon-monoxide in post detonation fumes at the face to 5 p.p.m. and 50 p.p.m. respectively within 5 minutes. In no case, quantity of air at any working place shall be less than 284 cu.m. per minute.

(b) Immediately before charging shotholes and again before firing the shots, tests for presence of inflammable gas shall be made with approved type methanometer at all places within a radius of 8 m. In goaf presence of gas shall be checked by using long probe. No shot hole shall be charged or fired if 0.1% or more inflammable gas is found.

(c) No person shall be allowed to remain on the return side of the faces where blasting is done till the concentration of NO_x and

carbonmonoxide in the return airway comes down to the limits specified above.

(d) Precautions against blasting fumes details in DGMS (Tech) Circular No. 4 of 1984 shall be observed.

3(a) The manager shall devise pattern of shot holes and it shall be strictly adhered to. Shot holes shall be slant towards goaf as proposed. Templates shall be provided for the purpose near the working places.

3(b)Charge of explosive in a round shall be so regulated that there is no adverse effect on roof and sides. Vibration studies shall be conducted to study the effect of blasting on strata.

4. Priming and blasting shall be done under the supervision of and in the presence of an assistant manager.

5. Long hole blasting shall not be practised at any place where two free faces are not provided. All shots charged in a round shall be fired simultaneously.

6(a)Explosive charges shall be directly initiated. In priming cartridges, detonator and detonating fuse/cord shall be inserted obliquely.

(b) In a shot hole, no part of detonating fuse/cord shall project beyond stemming material.

(c) In a shot hole jointing or overlapping of detonating fuse/cord shall not be done.

7. The charge of an explosive in a shot hole shall be evenly distributed with the help of dummy spacer cartridge and not more than 2 kg of explosive in an shot hole when used with spacers and approved type of detonating fuse/cord shall be used. Where spacers are not used, explosive charge in a shot hole shall not exceed 1 kg and no detonating fuse/cord shall also be taken into consideration for calculating explosive charge in a shot hole, (i.e., 200 m length of detonating fuse/ cord may be considerd equivalent to 5 kg of high explosives).

8. No shots shall be fired at any place unless the place itself, the adjoining goaf and all accessible places including the workings in top section if any within 90 m thereof have been treated effectively with water sprays. Suitable compressed air

operated mist-sprayer shall be installed close to the place of firing shot and shall be kept in operation before, during and after firing of shots.

9. While using Uniring explosives, detonation cord, etc., following shall be complied with :

(i) The entire detonating cord shalll remain within the shot hole inbye of the stemming material and the detonator and the detonating cord must be in contact with each other throughout the length of the detonator inside the primed cartridge.

(ii) For approved explosives cartridge diameter being 32 mm, the hole diameter shall be atleast 42 mm to accommodate a 6 mm diameter detonating cord.

(iii) The detonating cord shall be bound with the explosive cartridges using suitable type of adhesive tape.

(iv) For easy and proper charging of long holes, interlocking type of cartridges and spacers shall be used.

(v) Rataining raings may be used to hold the column charge in position.

(vi) A clay plug shalll be used at the end of each shot hole before charging and approved type water ampules may be used in shot holes.

(vii) After charging the shot holes compressed air generated mist sprayers strategically located close to the face shall be kept on.

(viii) The fumes generated by Uniring explosive (till date only one approved for long hole blasting) being more than normal, no person shall be sllowed to approach the face after blasting without waiting for atleast 20 minutes.

(ix) If any spacer is used, the distance between the cartridge shall not be more than 50 cm.

(x) Shot holes shall be drilled according to a ring pattern slopping at an angle of about 30 degrees to 40 degrees from vertical towards the goaf.

(xi) No delay action detonator shall be permitted and only one ring of shot holes shall be blasted in one round after the circuit has been tested with an ohm meter.

(xii) Only direct initiation would be permissible and the detonator shall be put on the outermost cartridges.

(xiii) Blasting shall be done in between the shifts when minimum number of persons are present below-ground.

(xiv) Strata control parameters and vibrations generated due to blasting shall be monitores by a scientific institute and a quarterly report about the performance of the system shall be sent.

Question 167 : With the help of neat sketches discuss the drilling and charging pattern for auger blasting.

Answer :

Blasting with Auger Drill

The principle related to this method is leaving of large diameter uncharged holes which act as free face for blasting of subsequent charged holes.

Drilling and charging pattern for auger blasting has shown in the Fig.73.

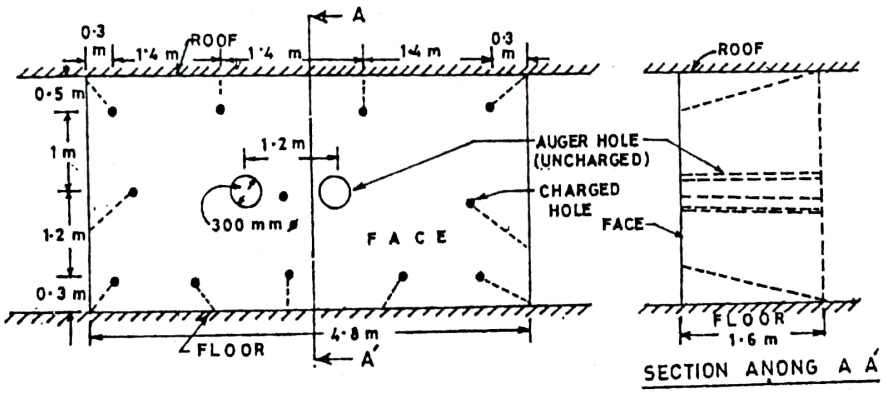


Fig. 73 Drilling Pattern for Auger Blasting

Here auger makes 2 holes of 300 mm size in the middle of the face. The details of blasting parameters are as following :

No. of holes : 12, No of auger holes : 2

Charge per hole-Top row : 450 gms, Middle row : 600 gms, Bottom row : 600 gms

Total charge/blasting : 6.6 kg

Middle and bottom rows are blasted in first round where as top row is blasted in 2nd round.

Explosive Type P1, Detonator Instantaneous copper electric.

Maximum charge per delay : 4.8 kg.

Average coal/blast : 18 - 20 tonnes.

Powder factor : 2.7 - 3.0 te/kg.

Average pull obtained : 1.1 - 1.15 m

Question 168 :

Explain the following :

a) Drilling pattern (b) Cut holes (c) Easers (d) Trimmers

What are the main drilling patterns which are applicable in stone drifting?

Answer :

Drilling Pattern in Stone Drifts

The shot holes in a stone drift are arranged in a particular form or pattern is termed as 'Drilling Pattern'. In a drilling pattern holes are generally divided into three groups e.g. cut holes, easers and trimmers.

Cut holes : Shot holes in this group are generally longer (approximately 15 cm) than the shot holes of the other group. These holes are fired first to create a free face for the shots of easers. Since these holes first make the opening in the face, they are prime responsible for the depth of pull.

Easers : The shot holes of this group are placed in the drift around the cut holes in two or more rings depending on the cross-sectional area. These holes ease the burden between the succeeding shot holes to enlarge the excavational area of the drift.

Trimmers : The shot holes of this group are placed around the easers which are fired at the last to make the final shape of the drift.

There are various types of 'drilling patterns' available for stone drifts and types of pattern are termed based on the type of cut holes. The main 'drilling patterns' are as following :

- Cone or pyramid or diamond cut
- Wedge cut
- Fan cut
- Drag or draw cut
- Burn Cut

Each hole in a drift may cover an area 0.25 to 0.6 m^2 depending on

the strength of rock whereas depth of pull generally varies from 1.5 to 3 m depending on the drilling pattern. The depth of pull depends on the cross-sectional area of the drift, diameter, length and angle of the shot holes. Burn cut gives greater depth of pull.

Question 169 : With the help of neat sketches explain the following :

a) Cone/Pyramid/Diamond Cut

b) Burn cut/Cylinder cut and Coromant cut

c) Draw Cut/Drag cut

Answer :

Cone/Pyramid/Diamond Cut

In this type of cut four or six shot holes are drived at the middle of the face which converges at the end to form either a cone, or a pyramid or a diamond shape and the holes are approximately 15 cm more in length compared to the other group of shot holes (Fig. 74). Maximum

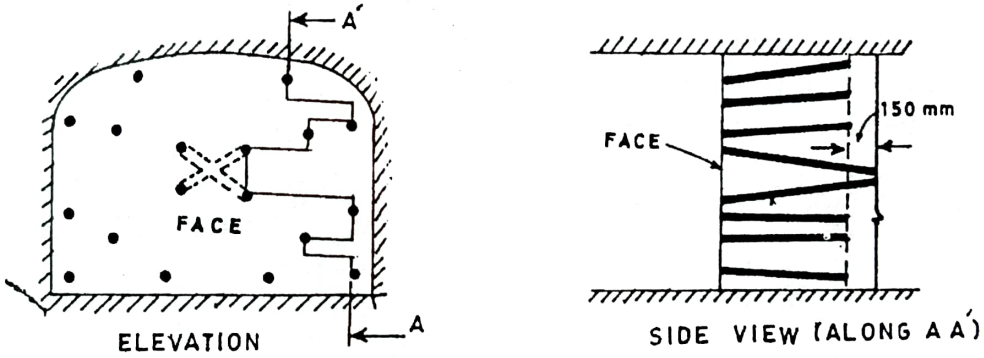


Fig. 74 Cone/Pyramid/Diamond Cut

concentration of charge is at the apex of these cut holes which are fired first to create a free face for the rest of the shots which are fired next with the delays. These type of cuts are generally used for blasting hard rock mass.

Burn cut/Cylinder cut

A cluster of parallel sot holes are drilled at right angles to the face to blast out a cavity in the centre of the heading. Some of the holes are charged while the others are kept empty and the blasting takes place towards these empty holes. The function of the empty holes is to provide a free face for reflection of shock waves so that the rock is shattered and subsequently blown out by the escaping gases. There is a specific geometrical relationship between the diameter of the empty hole and the

spacing between the centres of the empty hole and the charged hole in a given rock, which gives the essential condition of free breakage. Fig. 75 shows a burn cut pattern. With spacing less than 1.5 times the diameter of the empty hole, the blasting is expected to be clean while with higher spacing the breakage would tend to be uneven. With a spacing in excess of twice the diameter of the empty hole, there is a likelihood of plastic deformation and burning of rock.

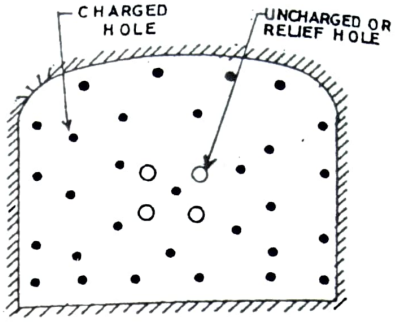


Fig. 75 Burn Cut

The optimum relationship between the diameter of empty hole and the spacing will vary with the nature of rock. There is obviously a greater degree of flexibility if the rock is brittle.

The above requisite of parallel hole blasting has important practical significance. If the diameter of the empty hole is 32 mm, which is the most common size of the shot hole used in development headings, the spacing will have to be as low as 48 mm between centres, i.e., leaving a rock web of only 16 mm between the shot holes, for satisfactory breakage. This is not feasible in practice unless rigorous precautions are taken to drill the holes accurately. Therefore, in practice, the spacing is perforce increased and this in turn. lead to the necessity of greater concentration of the explosive charge with a view to overcoming the constricted conditions so created and obtain some degree of breakage. The blasting effort is further augmented under these conditions by firing the cut holes simultaneously. The rate of advance is naturally restricted and is generally of the order of 1.2 - 1.5 m or less depending upon the nature of the rock.

For achieving greater advance, it is, therefore, necessary to increase the diameter of the empty hole which, in turn, gives better scope for increasing the spacing between holes without jeopardising the conditions of the free breakage. There are a number of variations of shot-hole pattern employing large diameter empty hole. Some of these are illustrated in Fig. 76 and 77.

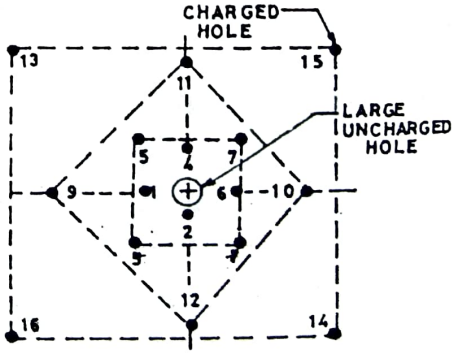


Fig. 76 Opening is Enlarged Cylindrically

These are generally termed as 'Cylinder' cut patterns. 'Coromant cut' a type of cylinder cut pattern is illustrated in Fig. 78.

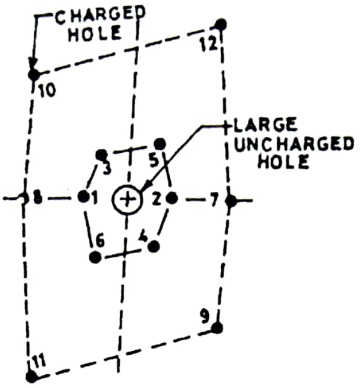


Fig. 77 Opening is Enlarged Spirally

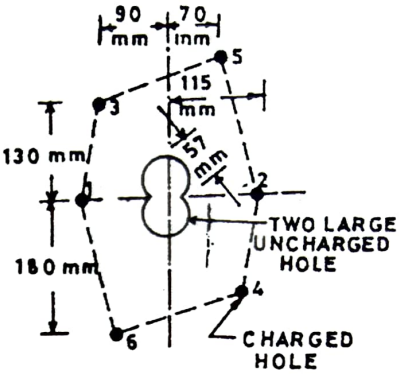


Fig. 78 Coromant Cut

The blasting is performed in such a way that as the charges in the first, second, and subsequent holes detonate in sequence, the opening is successively and uniformly opened in its entire length. Greater the diameter of the empty holes, greater is the depth of advance possible. The deviation of shot holes which assumes a serious proportion as the depth increases, is the only factor which ultimately limits the depth of the advance. The effect can, however, be overcome by increasing the diameter and/or number of empty holes. This type of cut is most suitable for hard, brittle, homogenous rocks but less effective in fractured rocks. With this pattern longer holes are drilled which give larger pull. In this type of cut drilling is easy and less time consuming compared to the angled hole drilling and smaller drifts can nicely be driven. Low power explosives are used in this type of cut with sequential blasting.

Draw Cut/Drag Cut

This type of cut is most suitable for the laminated rocks for 'controlled blasting' in the drivage of smaller cross-sectional area to break the rock along the cleavage plane. More number of shot holes are required in this case compare to the other pattern in the same cross-sectional area (Fig. 79)

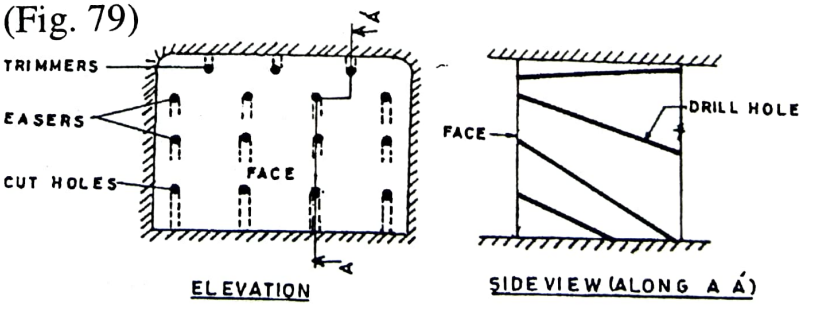


Fig. 79 Draw Cut

Question 170 : Write short notes on coromant cut.

Answer : It is a type of parallel hole cut. Two larger diameter (57 mm diameter) holes are drilled with the help of a template in such a manner that they will form an '8' shape. These holes are kept empty to act as a free face for the cut holes (drilled with the help of a template) around it (Fig. 78). So initiation of holes no. 1 and 2 with the help of delay detonator (direct initiation system) will give better fragmentation. The detonators may be placed in the charge of other holes in inverse initiation system. This type of cuts are very much useful for blasting in hard rock formation.

Question 171 : Why shall charging of different types of explosives in the same shothole be avoided? Why is inverse initiation practised in the solid blasting? What would be the basic safety features of an exploder to be used in the underground gassy degree-III coal mines?

Answer : Explosive characteristics like velocity of detonation, strength, water resistance, density, sensitivity, detonation pressure, stability, resistance to freezing, fume quality, cooling effects, etc. varies from explosive to explosive due to the different chemical composition. So use of particular type of detonator for a particular type of explosive may not cause initiation or cause delay initiation to the other type of explosives if they are inserted into the same hole. Hence, it would be better if same type of explosive is charged in a particular hole. For efficient transmission of V.O.D., the number of cartridges in a hole shall be as minimum as possible. In inverse initiation the C.J. plane is situated in the inbye (bottom) side of the blasthole. Hence after initiation of the primer charge the self propagating exothermic reaction moves through the explosives in the direction of the collar of the hole. The reaction will produce shock wave and various gases will exert tremendous pressure during expansion. In case of direct initiation, if the stemming is improper, then there will be huge amount of pressure energy losses through the loopholes of the stemming column. When sufficient pressure energy will be developed at the time of partial reaction of explosive there may be the chances of blowing out of the stemming material along with or without partially fragmented rock by the gas pressure since stemming length is the weakest point in the blasting block. The unexploded charge in this case will remain in the socket which is a very dangerous thing from the safety point of view. But in case of inverse initiation the

probability of occurrence of such phenomenon are practically very less. Hencc inverse initiation will enchance blasting efficiency and improve safety. In inverse initiation there will be also very less chances of accidental coming out of the detonator from the primer during stemming.

The basic safety features of an exploder to be used in the underground gassy degree-III coal mines are that circuits shall be intrinsically safe and efficient. Resistance of the circuit shall be very low and shall incorporate efflcient cut-off systems for power. The total circuits shall be housed in a flame proof enclosures and to avoid explosion due to ignition of gas the duration of supplied current by the exploder shall not exceed 4 milli-seconds.

Question 172 : Give some general safety guidelines regarding the drilling and firing of shot holes.

Answer :

The general safety guide lines are as follows

Drilling and Firing of Shotholes

1. The explosive cartridge should be at least 3 mm smaller in diameter than the shotholes.
2. No shothole shall be charged before it is thoroughly cleaned and its direction, where practicable, be marked at a convenient place.
3. Priming should be done just prior to blasting and near the face except in case of wet workings where it has to be done earlier. Detonators once inserted into a priming cartridge shall not be taken out.
4. The charge in any shot-hole shall consist of one or more complete cartridges of the same diameter and the same type of explosive. No charge in a shothole is overcharged or under-charged.
5. No shot shall be fired by a fuse less than 1.2 metres in length.
6. Every shot-hole shall be stemmed with sufficient and suitable non-inflammable stemming so as to prevent the shot from blowing out. Only sand and loosely filled in, or soft clay lightly pressed home, or a compact mixture of sand and clay, or water, shall be used as stemming material. In no case shall coal dust be used for purposes of stemming.

7. In charging or stemming a shot-hole, no metallic tool, scraper or rod shall be used. and no explosive shall he forcibly pressed into a hole of insufficient size.
8. No shot shall be fired except in a properly drilled, charged and stemmed shot-hole.
9. Blasting gelatine or other high explosive shall not be lighted in order to set fire to fuses.
10. All surplus explosives shall be removed form the vicinity of a shot-hole before a light is brought near to it for the purpose of lighting the fuse.
11. As far as practicable, a shot shall be fired by the same shot-firer who charged it.
12. In any mine in which explosives other than gunpowder are used, every shot shall, if so required by the Regional Inspector, be fired electrically.
13. Except in a stone drift or a sinking shaft, not more than 10 shots shall be fired in any one round. Where more than six shots are to be fired in one round, they shall be fired electrically.
14. No shot-hole shall be charged except those which are to be fired in that round and all shot-holes which have been charged shall be fired in one round.
15. Where a large number of shots have to be fired, shot-firing shall, as far as practicable, be carried out between shifts.
16. No person shall attempt to remove any stemming or pull out any detonator lead or remove any explosive from a shot-hole either before firing or after a misfire or bore out a hole that has once been charged or tampered with empty holes or sockets.

Question 173 : What precautionary measures are to be adopted during the use of permitted explosives? Discuss.

Answer :

Precaution in the Use of Permitted Explosives

- No shot shall be fired in coal unless

  the coal has been undercut, overcut or sidecut; and

the length of the shot-hole is at least 15 centimetres less than the depth of the cut;

- No detonator, shall be used, unless it is an electric detonator with a copper tube;
- Where more than one shots are charged for firing, the shots shall be fired simultaneously;
- The aggregate charge in coal shall not exceed such permissible maximum charge, as specified hy the DGMS. Only an approved shotfiring apparatus must be used in conjunction with permitted explosives.
- If in a ventilating district presence of inflammable gas is detected in any place, no shot-hole shall be charge stemmed or fired in that place or in any other place situated on its return side till such place has been cleared off gas and declared safe.
- Immediately before charging a shot-hole or a round of shot-holes and again before firing the shots, the shot-firer shall carefully test for inflammable gas at all places within a radius of 18 metres of the place of firing.
- No shot-hole shall be charged if any break is found therein, or if inflammable gas is found issuing therefrom.
- If after charging a shot-hole, inflammable gas is found in any place within the prescribed radius, no shot shall be fired until the place has been cleared off gas and declared safe.
- No delay action detonator shall be used, except with the previous permission from the DGMS.

Question 174 : Give some general safety guidelines regarding the 'Electric Shotfiring'.

Answer : The general safety guidelines under the heading is as following :

Only the approved electrical apparatus should be used to fire the shots and the maximum number of shots fired at one time by the apparatus shall not exceed the number for which it is designed.

- No current from a signalling, lighting or power circuits shall be used for firing shots.
- The shot-firer shall

- retain the key of the firing apparatus in his possession throughout his shift;
- use a well-insulated cable of sufficient length to permit him to take proper shelter and in no case shall the cable be less than 18 metres long;
- before coupling the cable to the firing apparatus, couple up the cable itself to the detonator leads;
- take care to prevent the cable from coming into contact with any power or lighting cable or other electrical apparatus;
- take adequate precautions to protect electrical conductors and apparatus from injury;
- himself couple the cable to the firing apparatus and before doing so, see that all persons in the vicinity have taken proper shelter, and
- after firing the shots and before entering the place of firing, disconnect the cable from the firing apparatus.

When more than one shots are to be fired at the same time

- care shall be taken that all connections are properly made;
- all shots, if fired belowground, shall be connected in series;
- the circuit shall be tested, either for electrical resistance or for continuity, before connecting it to the firing apparatus. Such a test shall be made with an apparatus specifically designed for the purpose and only after all persons in the vicinity have laken proper shelter.
- The cable to the shot-firing apparatus shall be connected last;
- detonator of the same electrical resistance shall only be used.
- A length of 45 m (the danger zone) with minimum 2 right angle bends should be cleared of all men and well guarded.
- In the underground before firing a shot, area within a distance of 18 m from the shot must be wetted.

Question 175 : What are the main causes of misfires in electric shotfiring? How is a misfire to be handled?

Answer : The following may be the causes of misfires in electric shotfiring :

Wrong or loose connections of detonators, unconnected detonators, exposed ends of lead wires becoming dirty during connections, discontinuity in the cable, damage in the insulation of the cable, internally short circuited cable, current leakage, too many detonators in the circuit, failure of the exploder to reach the rated output, defective detonators, defective explosives, inadequate priming, etc.

Handling Misfires

- No persons shall re-enter the place uptill 5 minutes after the firing of shots.
- In the event of a misfire, the entrance or entrances to the working place shall be fenced off so as to prevent inadvertent access and no work other than that of locating or relieving the misfired hole shall be done therin until the misfire has been located and relieved.
- In the event of misfire, a second charge shall not be placed in the same hole.
- If the misfire contains a detonator, the leads or fuse thereof shall be attached by a string to the shotfiring cable or some distinctive marker.
- Except where the misfire is due to a faulty cable or a faulty connection and the shot is fired as soon as practicable after the defect is remedied, another shot shall be fired in a releiving hole which shall bo so placed and drilled in such a direction that at no point shall it be nearer than 30 centimetres from the misfired hole. The new hole shall be bored in the presence of shot-firer, preferably the same person who fired the shot.
- After a relieving shot has been fired, a careful search for cartridges and detonators, if any, shall be made in the presence of the shot-firer, amongst the material brought down by the shot;

  Provided that in the case of working below—ground if such cartridge or detonator is not recovered, the tubes into which the material is loaded shall be marked and further search to be made on the surface. As for as possible, the search for the

detonators and cartridges and the loading of any coal, stone or debris which may contain a dentonator, shall be carried out without the aid of tools.

- If misfired hole is not dislodged by a relieving shot, the procedure laid down in the point (5) and (6) shall be repeated. A misfired hole which cannot be dealt with in the manner so prescribed, shall be securely plugged with a wooden plug, and no person other than a shot-firer, an official or a person authorised for the purpose shall remove or attempt to remove any such plug.
- When a misfired shot is not found or when a misfired shot is not relieved, the shot-firer shall, before leaving the mine, give information of the failure to such shot-firer or official as may relieve or take over charge from him.
- The shot-firer of the next shift shall locate and blast the misfired hole, but, if after a thorough examination of the place, the place where the misfire was reported to have occured, he is satisfied that no misfire has actually occured, he may permit drilling in the place.

Question 176 : What are the types of misfires in electric shot-firing? How those are to be handled? Discuss.

Answer : There may be two types of misfires in electric shot firing which are as following :

- all detonators failing to fire.
- some of the detonators failing to fire.

All Detonators Failing to Fire

In this case the procedure for handling misfire is as follows :

After disconnecting the cable from the exploder and detonator leads, continuity and short-circuitings of the cable are to be checked with an ohmmeter. In case of any defect in the cable, the fault is to be located and rectified. If there is no defect in the cable check the detonator connections visually for any fault. Reconnect the cable and test the circuit with an ohmmeter. In case of any discontinuity due to the faulty detonator, the Ohmmeter will read infinity. Locate the faulty detonator by the standard procedure of repeatedly dividing the circuit in halves and testing each part separately. After isolating the faulty detonator the rest of

the shots are to be fired. The misfired shot should be dealt with according to the procedure mentioned above. In case the circuit is in order the defect lies with the exploder and it should be rectified.

Some of the Detonators Failing to Fire

The causes of misfire may be due to current leakage, faulty exploder, number of detonators being more than the capacity of the exploder.

In case of current leakage one terminal of galvanometer is to be connected to the circuit and the other terminal with the ground. If there is current leakage the galvanometer gives a deflection. In case of faulty exploder or the number of detonators being more than the capacity of the exploder, the exploder should be checked.

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Chapter 4

Blasting Practices in Shaft Sinking and in the Underground Metalliferous Mines

Question 177 : Discuss drilling and blasting pattern of shafts to be sunk in the following cases :

a) Circular shaft in a coal measure strata

b) Circular shaft in a metalliferous mining deposit

c) Rectangular shaft in a metalliferous mining deposit.

Answer **(a_1)** *:* Circular shaft in a coal measure strata (excavated diameter of the shaft 7.8m).

i) Drilling for shaft sinking is generally done by the light weight pneumatically operated Holman or Silver - 3 Jack Hammer drill (consumption of air3 to 4 m^3/minute at a pressure of 5 to 7kg/cm^2). Since the excavated diameter of the shaft (Fig. 80) is 7.8 m, 6 drills operated by 6 drilling crew, (one in each drill) can nicely be employed for drilling holes. Two extra persons are to be engaged for removing the drill steel from the drilled holes and resetting in other places for drilling. Wet drilling is carried out to reduce the chances of jamming and the formation of dust.

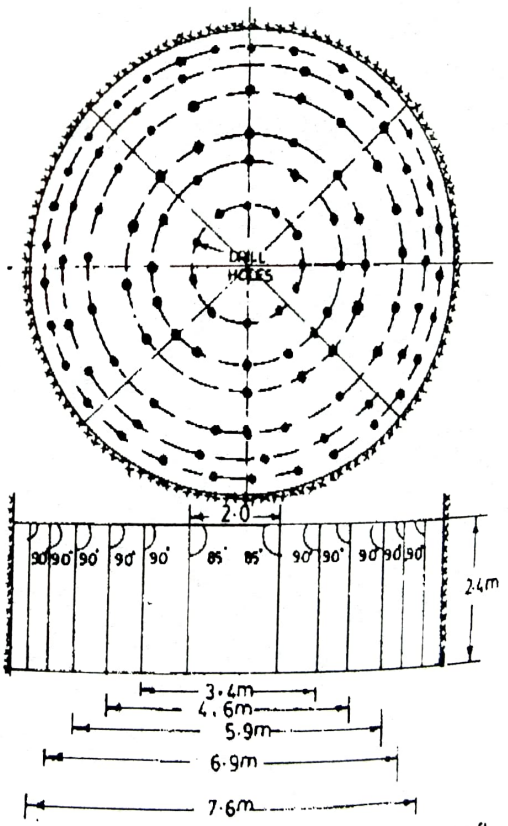


Fig. 80 A Typical blasting pattern of a shaft sinking operation in a coal measure strata (not to scale)

ii) The details of drilling

scheme is shown in Table - 39.

Table 39

Depth of hole (m)	Dia of hole (mm)	Total no. of Ring	Ring No.	Angle from horz. (Degree)	Dia. of ring (m)	No. of holes in ring	Total holes	Pull (m)
3.0	38	6	1st	85^0	2	11	108	2.2
2.4			2nd	90^0	3.4	12		
2.4			3rd	90^0	4.6	15		
2.4			4th	90^0	5.9	19		
2.4			5th	90^0	6.9	21		
2.4			6th	90^0	7.6	30		
Total time for drilling - 4 hours.								

iii) After drilling is over, all the holes are to be cleaned with conpressed air blow. All equipments and accessories are to be removed and scaffold shall be hoisted up at a distance of 40 to 50 m from the shaft bottom. In coal measure strata permitted explosives like Ajax-G, Unisex-G, etc. with a quantity of not more than 1 kg/hole shall be used. Shot holes are to be charged with the permitted explosives with inverse initiation using millisecond delay detonator. After charging the shot holes they are to be stemmed with moist sand. The sequence of delay is according to the ring. The firing circuit for the shaft can be used like parallel and series-parallel circuit depending upon the number of holes to be fired. Generally series circuit is used for firing small number of holes. Since in this case number of holes are to be fired is 108, either parallel or series-parallel firing circuit can be used for firing them by the mains firing system (440 or 550 volts mians). Schaffler explder can also be used.

Generally power factor obtained from sinking 0.66 to 0.77 m^3/kg of explosives.

Since length of the shothole is more, special permission from DGMS should be obtained to use more amount of charge per hole (say in this case 1.5 kg/hole).

Approximate cycle Timing -

Drilling	-	4 hours
Charging & Blasting	-	2 hours
Fume Clearance	-	1/2 hours
Firing inspection & dressing	-	1/2 hours
Mucking	-	18 hours
Floor clearance	-	2 hours
Total	-	27 hours

(a2) Another Example : The details of drilling scheme is shown in the Table -40

Table 40

Depth of hole (m)	Dia of hole (mm)	Total no. of Ring	Ring No.	Angle from horz. (Degree)	Dia. of ring (m)	No. of holes in ring	Total holes	Pull (m)
1.6	38	4	1st	69^0	1.8	8	54	1.45 to 1.5 m
			2nd	78^0	3.0	10		
			3rd	83^0	4.2	14		
			4th	93^0	5.6	25		

Excavated diameter of the shaft is 5.7 m in coal measure strata.

4 drill machines used simultaneously. No. of holes drilled per cycle-54

Cut used - Cone cut

Explosive used - Unisex - G

Detonators - Milli-sec. delay detonator is used in direct initiation system. The sequence of delay is according to the ring.

Charge per hole -750 gm each in 46 holes = 34.5 kg

900 gm each in 8 cut holes = 7.2 kg.

Total charge 41.7 kg or 42 kg approx.

Stemming material - Moist sand

Firing is done in series with Schaffler 750 exploder, cut off time is 4 milliseconds.

Powder factor - 0.76 to 0.83 m^3/kg of explosives.

Cycle Time

Drilling	-	1.75 hours
Charging & blasting	-	3 hours
Fume clearance	-	0.5 hours
Firing inspection and dressing	-	1.25 hours
Mucking	-	7.5 hours
Floor clearance	-	12 hours
Total	-	16 hours

The drilling works included the following :

- Centring of the shaft with a plumb and removing of plumb after centring.
- Vertically drilling of centre hole and marking of rings.
- Marking of holes at the periphery of the rings.
- Lighting of the shaft bottom and connection to a pump for dewatering of the floor if there is any water problem.
- Connection to each drill machine with compressed air (C.A.) hose through lubricator from C.A. manifold and also drill machines are to be connected with water manifold by a separate hose pipe.
- Drilling of holes in proper inclination at the marked points.
- To avoid jamming the holes are to be flushed regularly with water and C.A.
- After drilling is over each hole is to be plugged with wooden plug.
- Before charging remove all the materials lying on the shaft bottom.

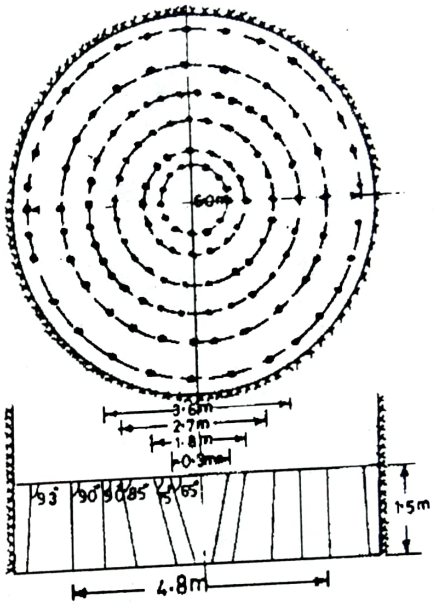


Fig. 81 A typical blasting pattern of a shaft sinking operation in a metalliferous mine (not to scale)

b) Circular shaft in a metalliferous mining deposit (excavated diameter of the shaft - 6.1m).

i) The drilling, etc. is the same as mentioned in Question 177 a, (i) except that 5 drill machines used here with 5 drilling crew (one in each drill machine). The details of drilling scheme is shown in Table-41.

Table 41

Depth of hole (m)	Dia of hole (mm)	Total no. of Rings	Ring No.	Angle from horz. (Degree)	Dia. of ring (m)	No. of holes in ring	Total holes	Pull (m)
1.5	38	6	1st	65^0	0.9	4	90	1.3
			2nd	75^0	1.8	8		
			3rd	85^0	2.7	12		
			4th	90^0	3.6	16		
			5th	90^0	4.8	21		
			6th	93^0	6.0	29		

ii) The details of blasting pattern is as following (Fig. 81)

Total tme for drilling - 8 hours

Cut used - cone cut

Life of the drill bit - 110 m drilling (approx.)

iii) After drilling is over, holes are to be cleaned with compressed air blow. All equipments and accessories are to be removed and scaffold shall be hoisted up at a distance of 40 to 50 m from the shaft bottom.

(iii) After drilling is over, holes are to be cleaned with compressed air blow. All equipments and accessories are to be removed and scaffold shall be hoisted up at a distance of 40 to 50 m from the shaft bottom. 60% special gelatine cartridges are used to blast the holes with the help of millisecond delay detonator in inverse initiation. The sequence of delay is according to the ring. After charging the holes thay are to be stemmed with moist sand. In this case either parallel or series-parallel firing circuit can be used for firing them by the mains

firing system (440 or 550 volts mains). Stemming material used is the moist sand. Generally powder factor obtained- 0.55 to 0.66 m^3/kg of explosives.

Approximate cycle timing

Drilling	-	8 hours
Charging and blasting	-	2.5 hours
Fume clearance	-	0.5 hours
Firing inspection and dressing	-	1 hours
Mucking	-	12 hours
Floor clearance	-	2 hours
Total	-	26 hours

(c) Rectangular shaft in a metalliferous mining deposit (size 5.5 m x 4.8 m)

i) The drilling, etc. is the same as mentioned in Question 177 b (1).

ii) The blasting pattern is shown in the Fig. 82. The depth of the

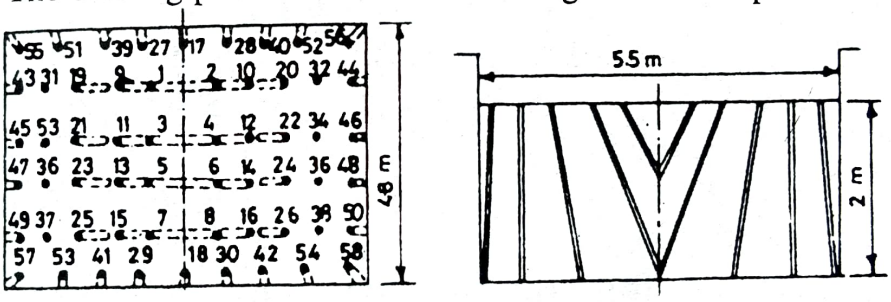


Fig. 82 Showing the details of typical blasting pattern in rectangular shafts

round drilled is 2 m and the pull obtained is about 1.8 m. The details of drilling pattern is shown in Table-42

Table 42

Holes Sl. No.	Dia. of drill hole (mm)	Delay No.	No. of holes	Charge/hole (kg)	Total No. of holes
1-8	38	0	8	1	58
9-18		1	10	2	
19-30		2	12	1.8	
31-42		3	12	1.8	

Table : 42 (*continued*)

Holes Sl. No.	Dia. of drill hole (mm)	Delay No.	No. of holes	Charge/hole (kg)	Total No. of holes
45-54		4	12	1.8	
55-58		5	4	2.0	
Total charge 100 kg					

Cut used - Wedge cut.

iii) Cleaning and charging of holes, firing etc. are the same as mentioned in Question 177b (iii)

(However half second delay detonator and Beethoven Dynamo condenser exploder (for firing 100 shots) or Schaffler 750 (for firing 100 shots) can also be used in this case). Stemming material used is the moist sand. Powder factor obtained - 0.47 to 0.5 m^3/kg of explosive.

DRIFT, RAISE, WINZE IN METALLIFEROUS MINING DEPOSITS

Question 178 : Discuss drilling and blasting techniques showing typical blasting pattern of the following in an underground metalliferous mine -

a) A Drift (size : 2.3 m wide and 3 m height)

b) A Drive (size : 2.2 m wide and 2.25 m height) in a very hard rock formation.

c) A drift (size : 2 m wide and 2.5 m height) in the hardest rock formation.

***Answer (a) :* A drift (size : 2.3 m wide and 3 m height)**

i) The typical wedge cut blasting pattern shown in the fig 83.

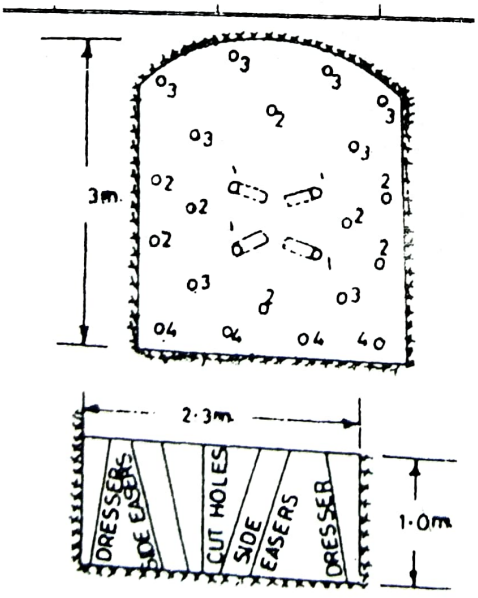


Fig. 83 A typical wedge cut blasting pattern in a drift in a metalliferous ore deposit (not to scale)

Here 3 types of holes are drilled in the wedge cut blasting pattern - (i) Cut holes (ii) Easers (iii) Trimmers/Dressers.

The details of drilling scheme is shown in Table - 43.

Table 43

Type of holes	Dia. of hole (mm)	No. of holes	Delay detonator used	Length of holes (m)	Average pull (m)
Cut holes	34	4	1	1.2 m	0.85 m
Easers	34	8	2 & 3	1.2 m	
Trimmers/ Dressers	34	12	2, 3, 4	1 m	

ii) Charging of holes

It is done by 80% or 90% S.G., Dia. of cartridge - 25 mm

Length - 250 mm

Weight of each cartridge 140 gm

Volume of ore blasted per round - 5.86 m^3

Tonnage of ore blasted per round = 5.86 x 3.2 = 18.768 tonne (Assumed density of ore - 3.2 tonne/m^3)

iii) Detonators are connected in series and blasted by Beethoven Dynamo Condenser type exploder (6 μF Condenser to a potential of not less than 1200 volt, D.C.)

iv) The stemming materials used is the mixture of sand and clay in 3:1 proportion (also a little amount of calcium chloride is mixed) and cartridge of 25 mm dia and 150 to 200 mm length is formed by drying. The length of the stemming is usually 66% of the length of the hole. The first 2 or 3 stemming cartridges near the charge should be tamped very lightly and the rest should be tamped hard with the help of a wooden stemming rod. The details of drilling and charging schemes are shown in Table - 44.

Table 44

Type of holes	No. of hole	Cartridge/ hole	Total no. of cartridges	Weight (kg)	Powder factor, tonne/ kg of explosive
Cut holes	4	3	12	1.68	$\frac{18.768}{8.4}$ = 2.23
Easers	8	3	24	3.36	
Trimmers/ Dressers	12	2	24	3.36	
			Total charge 8.4 kg		

Before charging operation the holes are to be cleaned for dust and water and the prime cartridge is generally placed in the hole in inverse initiation system.

The bit dia. of the hexagonal drill rod : 33 mm

Length - 1 m, 1.5 m etc.

Rate of penetration varies according to the hardness of rock- generally it is 10 cm to 20 cm per minute in hard rock formation depending upon the air pressure.

The drill machine used is usual silver - 3 Jack hammer, with air consumption - 3 to 4 m^3/minute at a pressure of 5 to 7 kgf/ cm^2.

v) Cycle of Operation

Drilling - 4½ hours
Mucking - 3 hours
Charging and blasting - 1 hours

Total 8 hours and 30 minutes

Drilling crew
Mistry - 1
Driller - 1
Helper - 1
Total - 3

Mucking and transporting of tubs - 6 persons
Supervisor - 1

(b) A drive (size 2.2 m wide 2.25 m height) in a very hard rock formation.

i) The typical wedge cut blasting pattern with a chip cut is very

useful in this case which is shown in the figure - 84.

Here 4 types of holes are drilled in the blasting pattern-

(i) Chip cut (ii) Cut holes (iii) Easers (iv) Trimmers/ dressers.

ii) Charging of holes

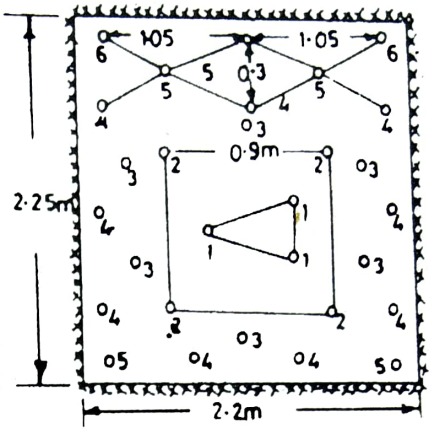
It is done by 80% or 90% S.G., Dia. of cartridge - 25 mm

Length - 250 mm and weight of each cartridge - 140 gms.

Volume of ore blasted per round - 7-17 m^3

Tonnage of ore blasted per round - 23 tonne

(Assumed density of ore is 3.2 tonne/m^3)



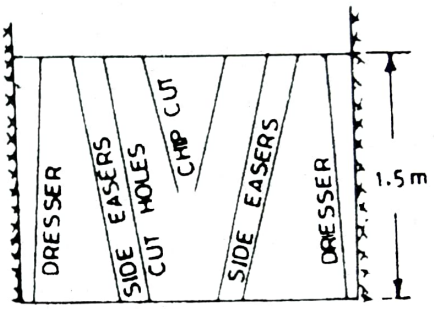


Fig. 84 A Typical wedge cut blasting pattern with chip cut in a drive in a metalliferous ore deposit (not to scale)

The stemming, charging of holes, drill bit and drill machine, manpower etc. are the same as mentioned in question No. 178 a (iii), (iv) and (v).

The details of drilling and charging of holes are shown in the Table-45 and Table-46.

Table 45

Type of holes	Dia. of hole (mm)	No. of holes	Delay detonator used	Length of holes (m)	Average pull (m)
Chip cup	34	3	1	1	1.45
Cut holes	34	4	2	1.7	
Easers (including top easers)	34	9	3, 4, 5	1.7	
Trimmers/ Dressers	34	13	4, 5, 6	1.5	

Table 46

Type of holes	No. of hole	Cartridge/ holes	Total no. of cartridges	Cartridge Weight (kg)	Powder factor, tonne/ kg of explosive
Chip cup	3	3	9	1.26	$\frac{23}{11.48} = 2$
Cut holes	4	5	20	2.80	
Easers	9	3	27	3.78	
Trimmers/ Dressers	13	2	26	3.64	
			Total charge 11.48 kg		

Cycle of Operation

Drilling - 7½ hours

Mucking - 4 hours

Charging and blasting - 1½ hours

Total - 13 hours

(c) A Drift (size : 2 m wide and 2.5 m height) in the hardest rock formation

i) The typical burn cut blasting pattern shown in the fig. 85.

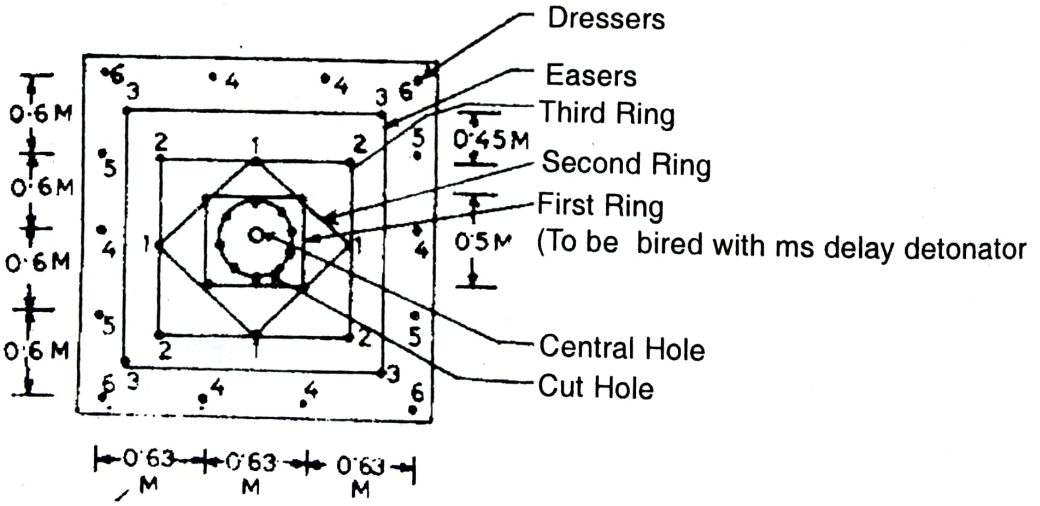


Fig. 85 Shows a burn cut pattern of blasting in a drift in a metalliferous ore deposit

Here the following types of holes are drilled

ii) **Charging of holes**

It is done by 90% S.G., Dia of cartridge - 25 mm

Length - 250 mm

Weight of each cartridge - 140 gm

Volume of ore blasted per round - 7 m^3

If density of ore is 3.2 tonne/m^3

then tonnage of ore blasted per round - 22.4 tonne

The stemming, charging of holes, drill bit and drill machine and manpower, etc. are the same as mantioned in question no. 178 a (iii), (iv) and (v).

Cycle of operation

Drilling	-	10½ hours
Mucking	-	5 hours
Charging & Blasting	-	1½ hours
	Total -	17 hours

The details of drilling and charging of holes are shown in the Table - 47 and 48.

Table 47

Type of holes	Dia. of hole (mm)	No. of holes	Delay detonator used	Length of holes (m)	Average pull (m)
Central hole	76	1	-	1.5	1.4
Cut hole	34	9	0	1.5	
First ring	34	4	0	1.5	
Second ring	34	4	1	1.5	
Third ring	34	4	2	1.5	
Easers	34	4	3	1.5	
Trimmers/ Dressers	34	14	4, 5, 6	1.5	

Table 48

Type of holes	No. of hole	Cartridge/ hole	Total no. of cartridges	Weight (kg)	Powder factor, tonne/ kg of explosive
Cut hole	9	5	45	6.3	

Table 18 (*Contd.*)

Type of holes	No. of hole	Cartridge/ hole	Total no. of cartridges	Weight (kg)	Powder factor, tonne/ kg of explosive
First ring	4	4	16	2.24	
Second ring	4	3	12	1.68	
Third ring	4	3	12	1.68	$\frac{22.4}{17.5}$ = 1.28
Easers	4	3	12	1.68	
Trimmers/ Dressers	14	2	28	1.68	
Total charge 17.5 kg					

Question 179 : With the help of a neat sketch of blasting pattern discuss drilling and blasting system of a morern larger size drifting in stone.

Answer : Fig. 86 shows blasting pattern with typical burn cut system in a modern drifring.

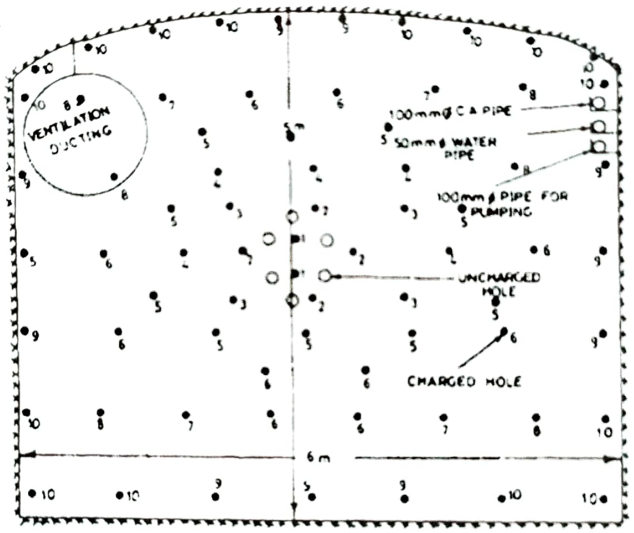


Fig. 86 Shows a modern drifting with burn cut system of drilling and blasting

Size of the drift - 6 m width and 5 m height

Gradient of the drift - 1 in 50.

Size of the drill hole - for charged hole – 51 mm dia.
Total No. 72,

for uncharged hole - 76 mm dia
Total no. 6

Depth of the hole - 3.5 m

Drilling is done with a 2 boom electro hydraulic jumbo drill. For faster progress it is always adviseable to use high capacity electrohydraulic jumbo drill along with wheel loaders for quick loading and because of larger dimension of the drift, 15 to 20 m^3 tub capacity trucks can efficiently be used for muck disposal.

With efficient drill machine the drilling time for the 72 nos. charged holes and 6 nos. uncharged holes will take around 5 to 8 hours depending upon the hardness of rock.

Charging and stemming is generally carried out by a charge up truck fitted with pressurised ANFO loader (since most of the holes are blasted with ANFO) and 10 t crane fitted with basket. With the help of this crane and basket the shot-firer/charger can charge explosive and stem them very easily at a minimum time.

All the charged holes are fired with ANFO with Superdyne as a primer charge. The detonator used with superdyne is NONEL connected with detonating fuse and the primer is placed into the hole in inverse initiation system.

Charge per hole (except trimmers) - 3.5 kg

Out of this 3.5 kg, Superdyne is 1 kg (40 mm dia. cartridge, two in number)

and ANFO – 2.5 kg

ANFO is charged into the holes with pressurised loader. Trimmer holes are charged with small diameter decoupled cartridges strung together and by providing good stemming at the collar of the hole. Charges are fired simultaneously after the lifter. This is done for the formation of smooth wall and better ground control.

Here 25 mm dia superdyne cartridges are used.

Total explosive consumption - $45 \times 3.5 = 157.5$ kg.

$27 \times 1.8 = 48.6$ kg

Total 206.1 = 206 kg

Pull per round 3.2 m

Volume of rock produced/round = 96 m^3

Powder factor = 0.46 m^3/kg of explosive.

Question 180 : Discuss blasting practices in a raise of size 2.3 m x 1.2 m.

Answer : 1. The blasting pattern with sequence of delay has been shown

in the fig. 87.

The cut pattern used - wedge cut pattern.

Here 3 types of holes are drilled in the wedge cut blasting pattern (1) Cut holes (2) Easers (3) Trimmer/Dressers.

2. Charging of holes

It is done by the 80% S.G. dia. of cartridge - 25mm

Length- 250 mm and weight of each cartridge - 140 gms.

In the footwall side trimmers/dressers- 3 cartridges are used per hole.

Volume of ore blasted per round - 3.86 m^3

If the density of ore is 3.2 tonne/m^3

Then tonne of ore blasted per round - 12.36

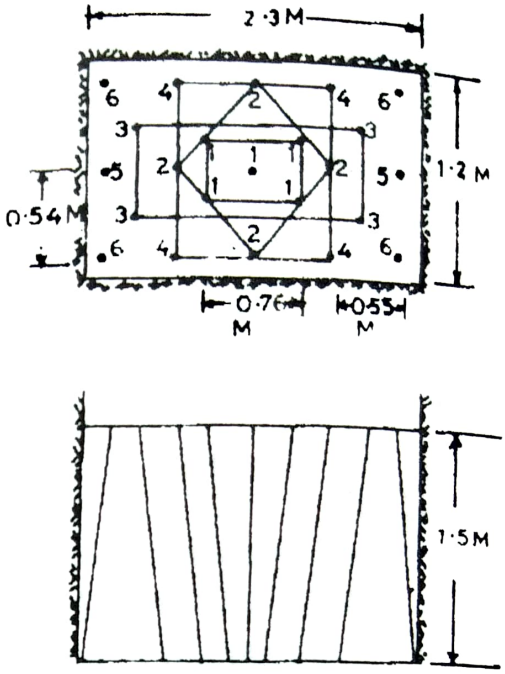


Fig. 87 Shows wedge cut pattern of blasting in a raise in a metalliferous ore deposit

3. Both the half second and milli-second delay detonators of ranging 0 to 10 may be used. Detonators are connected in series and blasted by the Beethoven Dynemo Condenser type exploder.

The details of drilling and blasting are shown in the Table-49 and 50.

Table 49

Type of holes	Dia. of hole (mm)	No. of holes	Delay number used	Length of holes (m)	Average pull (m)
Central hole	34	1	1	1.5	1.4
Cut hole	34	4	1	1.7	
Easers	34	6	2, 3, 4	1.7	
Trimmers/ Dressers	34	12	2, 4, 5, 6	1.5	

Table 50

Type of holes	No. of hole	Cartridge per hole	Total no. of cartridge	Weight (kg)	Powder factor tonne/kg of explosive
Central hole	1	4	4	0.56	$\frac{12.36}{9.38}$ = 1.32
Cut hole	4	4	16	2.24	
Easers	6	3	18	2.52	
Trimmers/ Dressers	12	2	2x7+5x3=29	4.06	
Total charge 9.38 kg					

4. The raise driven by drilling and blasting method can be done with the practice of the following methods :

- Open raising
- Two compartment raising
- Jora raise lift
- Alimak raise climber.
- Long hole drilling method etc.

Open Raising – The method is simplest and most commony used, wherein workers stand on a plalform made of timber planks, supported on prefabricated construction material like clamps, iron bar or bull horns. The clamps used for supporting the platform are made in standard lengths, out of old 30 lbs rails. Before each round is blasted the platform is dismantled. Access to face is made by rope ladders etc. The method is well suited for raises of moderate length (upto 6 m) and inclinations (40^0 to 60^0 with horizontal).

Two Compartment Raising– In this system a timber portion separates the raise in two compartments, major cross section being used by broken rock and the rest for placing of pipe ladders for access to the face. At times broken rock serves as platform for performing various operations. The ladders compartment is covered with wooden planks at the time of blasting. It is applicable in the driving of raises in most varied conditions, preferably steep ones.

Jora Raise Lift – In this method at first a pilot hole is drilled from

upper level to lower lavel and a cage is suspended by a wire rope through a pilot hole from the upper level. Originally the hoist used to be fixed in upper level necessitating a hoist operator with added provisions for communications. In improved version a jora type cage, housing a hoist within the cage itself and acts like a spider, climbs up and down along a hoisting rope suspended through the pilot hole. The number of holes to be drilled and survey work involved are considerably reduced due to the pilot hole which serves as burn hole as well as guide to direction and inclination and improves ventilation. The method is applicable to vertical and inclined raises upto 100 m length.

Raising by Alimak Raise Climber- Let the raise mentioned in the question has been driven with the help of alimak raise climber. It is essentially a self-powered rugged platfrom manoeuvrable for any height and angle, moving on a rail guide anchored to the roof of the raise. To carry men and material up and down a cage is attached to the platform, the later being powered by compressed air or electricity. The air is carried to the air motor by a hose round on a self-controlled hose reel kept below. The platform climbs along with pin-rack integral with the guide rails, available in sections wich are extended as the raise progresses. The guides are anchored to the walls by retractable expansion bolts. The guide rails are cast integral with four 1" pipes, two carrying compressed air, the third one for carrying water and the fourth for blasting cable and telephone cable, etc. An automatic air and water manifold supplies relevant connectin to the face through the pipes. During blasting a header plate with a water nozzle and vents for air to shower the face, is bolted to the top of the last guide and serves as protests. The blowing of air-water ventilates the face within a short period. While for inspection and scaling after blasting, the safety roof keeps a way the falling pieces of rock. A crew of 2 driller and 2 helpers take 3 drilling machines and accessories up. Keeping one machine as stand by, drilling is done with the rest two, stopers being preferred for raises steeper than 60^0 inclination. Once the drilling is over, blasting header plate replaces drilling header plate and drilling crew and equipment are withdrawn. Then a blaster and a helper assisted by one cage operator approach the face for charging. After charging, the cage is taken in the cross-cut corner so as to save it from being damaged by blasted fragmets. While blasting the air water lines are fully opened. The fume clearance will take 5 minutes if bottom of the

raise is in ventilation circuit otherwise it will take 30 minutes to 1 hour. After fume clearance, before going for inspection, the equipment is washed thoroughly. A 2 m guide is taken by the drilling crew after finishing scaling work. The guide is placed in the last guide and anchored with the help of expansion bolts inserting spacers if necessary and finally brackets. After every blast all the nuts of extended guide spacers, brackets and bolts must by checked for their tightness.

The stemming, charging of holes, etc. are the same as mentioned in question No. 178 a(iii).

Cycle of Operation

To drill 23 holes of average length - 1.5 m - 4 hours

(By two drill machines each operated by 1 driller and one helper.)

Charging, blasting and withdrawal of cage to the safe place	- 2 hours
Fume clearance, cleaning of walls and equipment etc.	- 1 hour
Inspection, washing and scaling of loose at face	- 2 hours
Extension and anchoring of 2 m guide and making connections etc. for the next round drilling	-2 hours
Total time	- 11 hours

Within the time of extension and anchoring of 2 m guide and also during drilling operation the whole muck will be cleaned by a rocker shovel placed at the cross cut connected the raise.

The method of raising by the Alimak Raise Climber is very economical for raises above 40 m in length and more than 40^0 inclination with horizontal.

Question 181 : Discuss down the hole drilling and blasting practices in Drop Raising.

Answer : **Down the Hole Drilling and Blasting Practices for Drop Raising**

Fig. 88 shows a schematic diagram of blasting pattern and practices of drop raising by the down the hole drilling and blasting methods in a

metalliferous mine. Here 5 long blast holes (length - 20 m, diameter of hole - 165 mm, inclination of holes - vertical, 4 corner blast holes and 1 vertical blast hole) are drilled downward by a crawler drill model ROC - 306 with the help of booster compressure (Model DG-418) pack of Atlas Copco make. The working pressure of the ITH rock drill is 6 to 18 bar and that of air motor is 6 bar. The ROC-306 and drill holes at an inclination from 65^0 to 90^0 upto a length of 100 m and 200 m for the diameter of holes 165 mm and 115 mm respectively. Compressive strength of the rock is 1000 to 1400 kg/cm^2 and that of ore is 400 to 600 kg/cm^2.

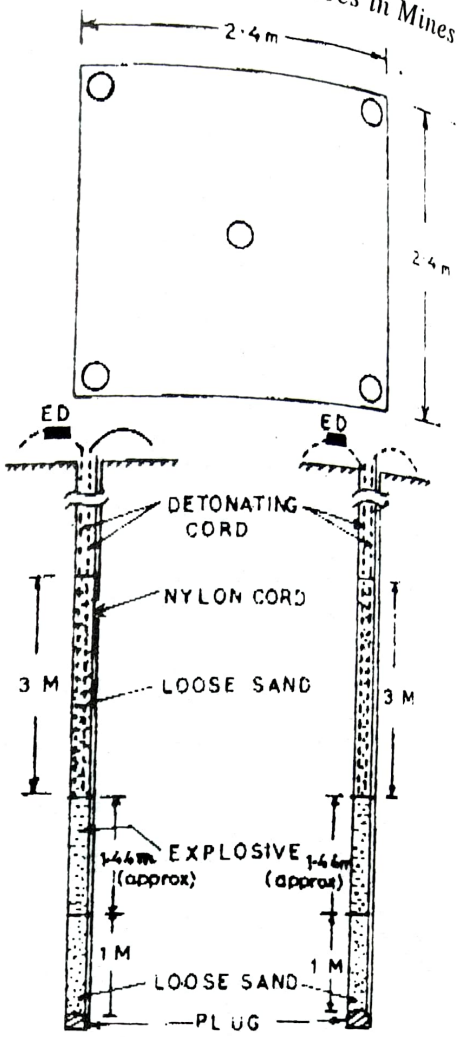


Fig. 88 Shows down the hole drilling and blasting for drop raising

Size of the raise - 2.4 m x 2.4 m

For charging the shot hole first the bottom of the hole is plugged with the help of a nylon cord and then stemming is done upto a length of 1 m above the plug. Above stemming 3 cartridges of either Indoboost or Toe-blast (125 mm dia. and approximately 480 mm length and 6.25 kg weight) is used. Above the cartridge stemming is done upto a height of 3m. The top most cartridge acts as the primer which is initiated by the cordtex detonating fuse through electric delay detonator placed near the blasthole collar (i.e. all the holes are blasted in sequence with delay intervals).

The average advance per round of blasting - 1.5 m.

Question 182 : Discuss blasting practices in a winze of size 2.8 m x 1.8 m.

Answer : 1. The blasting pattern with sequence of delay has been shown in fig. 89.

The cut pattern used - Wedge cut pattern.

Here 3 main types of holes are drilled - (1) Cut holes (2) Easers (3) Trimmers/Dressers.

2. Charging of holes

   It is done by the 80% S.G., Dia of the cartridge - 25 mm Length - 250 mm and weight of each cartridge - 140 gms

   Volume of ore blasted per round = 2.8 x 1.8 x 1.4 = 7.05 m^3

   If the density of ore is 3.2 tonne/m^3

   then tonnage of ore blasted per round - 22.58 tonne

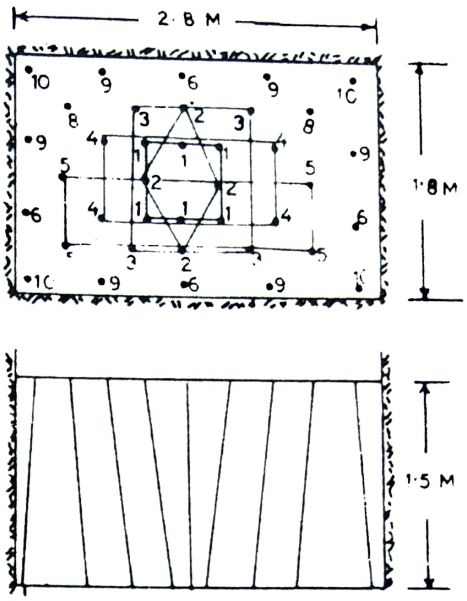


Fig. 89 Shows wedge cut pattern of blasting in a winze in a metalliferous ore deposit

3. Both the half second and milli-second delay detonators of ranging 0 to 10 may be used. Detonators are connected in series and blasted by the Dynamo Condenser type of exploder.

4. The stemming, charging of holes, drill bit, manpower etc. are the same as mentioned question no. 178 a (iv) and (v). Drill machine used is Silver - 3 drill machine which works at a pressure of 5 to 7 kgf/cm^2.

   Diameter of the compressed air pipe line is as following -

   Main line - 200 mm, branch line - 100 mm

   Secondary branch line near the face - 50 mm

   The drill machine is connected to this 50 mm dia. pipe range by means of a rubber hose.

The details of drilling and charging of the holes are given in the Table-51 and 52.

Table 51

Type of holes	Dia. of hole (mm)	Delay number used	No. of holes	Length of holes (m)	Average pull (m)
Cut hole	34	1	6		
		2	2	1.7	

Table : 51 (*Contd.*)

Type of holes	Dia. of hole (mm)	Delay number used	No. of holes	Length of holes (m)	Average pull (m)
Easers	34	2	2		1.4
		3	4		
		4	4	1.7	
		5	4		
		8	2		
Trimmers/ Dressers	34	6	4	1.5	
		9	6		
		10	4		

Table 52

Type of holes	No. of hole	Cartridge per hole	Total no. of cartridge	Weight (kg)	Powder factor, tonne/kg of explosive
Cut hole	8	5	40	5.6	
Easers	16	4 cartridges each for 10 holes	40	5.6	$\frac{22.58}{18.2}$ = 1.24
		3 cartridges each for 4 holes	18	2.52	
Trimmers/ Dressers	14	3 cartridges each for 4 holes	12	1.68	
		2 cartridges each for 10 holes	20	2.80	
				Total charge 18.2 kg	

5. Before commencement of winze a raise of 7 to 8 m height is to be driven at the roof of the drive to accommodate the hoist chamber, where a small winding engine (motor power 15 to 20 kw) to be established. The pulley (250 to 300 mm dia) of

the hoist is generally located at a height of 6 m in that raise with the help of two rail girders.

The diameter of the hoisting drum is generally 0.6 m and the capacity of the hoppit is 0.5 to 1 tonne.

Mucking is done with the help of this hoist.

Number of persons required for mucking, hoisting and transporting of bandies - 6 persons along with a supervisor.

6. Air current in the winze is supplied by an auxiliary fan of motor power – 5 kw.

Question 183 : Give an idea about the blasting practices in a conventional Breast Stoping in ore deposit of 1.15 m thick dipping at 30^0.

Answer : In this stoping method levels are developed at 37 m vertically apart. Length of the stop face 66 to 70 m from level to level. For ventilation and stoping, raises are constructed in payable zones, at an interval of about 100 m. Here the stope face is carried along full dip.

Drill machine used/shift – 6

Length of hole – 1.5 m

Dia. of hole - 34 mm

Burden of holes - 500 mm

Spacing of holes - 500 mm

Three holes are drilled in one line.

Approximate pull - 1.4 m

No. of holes drilled/drill machine/shift – 60 nos.

So 6 drill machine drilled blast hole/shift = 6 x 60 = 360 nos.

Explosive used - 60% S.G.

Cartridge dia - 25 mm

Amount of explosive used/hole – 350 gms

Total quantity of explosive used/shift = 0.35 x 360 = 126 kg

Amount of ore produced by blasting 360 nos. holes - 96.6 m^3 of ore

If the density of ore is around 3.2 tonne/m^3 then

amount of ore produced/shift = 309 tonne

Powder factor - $\frac{309}{126}$ = 2.45 tonne/kg of explosive

Amount/hole - 0.858 tonne

Drill machine used - Silver-3 drill machine

Before charging the holes, they are to be cleaned for dust and water, and the prime cartridge is generally placed into the hole in inverse initation system. The stemming materials used is the mixture of sand and clay in 3:1 proportion (also a little amount of calcium chloride is mixed) and a cartridge of 25 mm dia, and 150 to 200 mm length is formed by drying. The length of the stemming is usually 2/3rd of the length of the hole. The first 2 or 3 stemming cartridges near the charge should be tamped very lightly and the rest should be tamped hard with the help of wooden stemming rod.

The milli-second delay detonators are connected in series and blasted sequentially in groups by a Dynamo Condenser type exploder. Total 360 holes are divided into number of groups and holes in each group is blasted at a time.

Daily Manpower

Driller	-	18
Blaster and helper	-	16
Mucker and trammers	-	84
Timberman	-	100
	Total -	218

Question 184 : Discuss the drilling and blasting practices in the most modern highly mechanised cut and fill stoping.

Answer : A schematic layout of the most modern highly mechanised cut and fill stoping has been shown in the Fig. 90.

The thickness of ore is around 5 m and dipping at 80^0 occured in the hilly terrain. Opening of the deposit is done by the adits. Strike length - 2.5 km. The distance between two sub-level is 20 m with a cross section of each sub-level 5 m width x 5 height. This type of cut and fill method totally eliminates conventional rail equipments by introducing rubber tyred mobile equipments which can move freely to any production or development faces in any suitable area of the ore body via footwall ramp (1 in 6 in gradient) which are connected to each sublevel. This will provide efficient utilisation of every machine, nicely control the grade of ore and improve the overall productivity. These ramps will provide better access to the ore body, development heading, stopes, etc. as required by

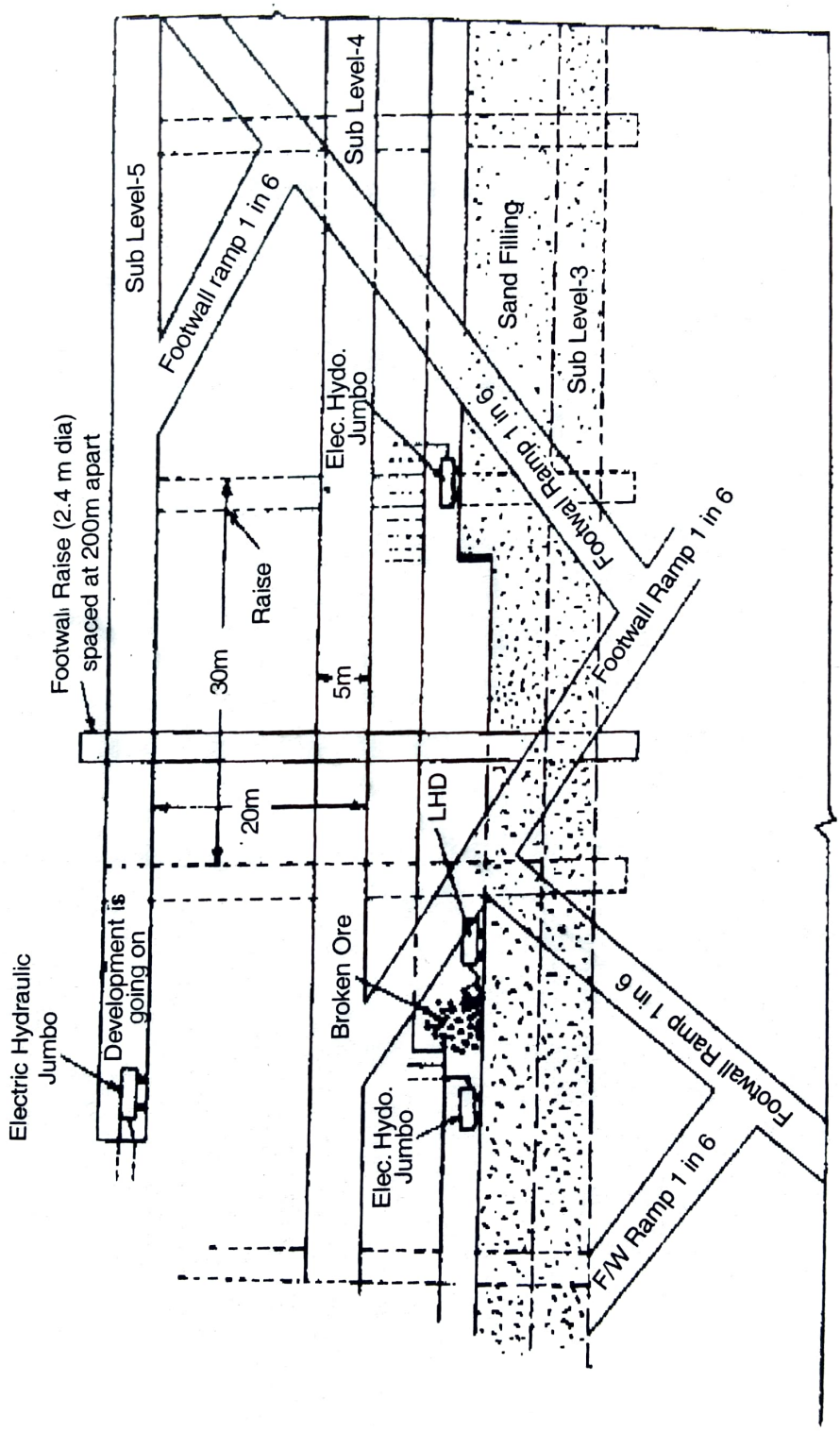


Fig. 90 Shows a section of the most modern highly mechanised cut and fill stoping

providing greater flexibilily rationalising the movement of the equipments.

Drilling in the stoping and in the development faces are done by the tyre mounted 2 boom Electro-Hydraulic Drill Jumbos. The length of the drilled blast hole in the stope is 4.1 m to extract a 4 m lift from the back of the strike drive and after taking out the ore the sand is filled in the stope within 4 m from the backs. From the stowed sand filled floor another tyre mounted 2 booms Electro-Hydraulic Drill jumbos will drill another 4 m lift for continuing the cycle (Fig. 90). Broken ore is cleaned by LHD (capacity of the bucket 2.3 m) into the ore pass.

Drilling in the stope-

It is done by the tyre mounted electro-hydraulic drill jumbo.

The dia. of the hole 51 mm.

Bit used – Button bit

Spacing of holes – 1.1 to 1.2 m

Burden of the holes – 0.9 to 1 m.

Length of the each blasthole – 4.1 m

Average drilling shift/drill jumbo – 74 to 90 m

Average number of drilled blast hole/shift/drill jumbo – 18 to 22.

(Say on an average 20 nos.)

Drilling is done by one driller with one helper per drill jumbo.

Life of the drill bit which depends upon the hardness and mineralogy of the ore, is around 250 to 350 m of drilling.

Blasting

Explosive used in the hole - ANFO with 60% S.G. as primer initiated with Nonel milli-second delay detonator and blasted with detonating cord.

Quantity of charge per hole – 5 kg including 1 kg of 60% S.G.

ANFO is loaded into the holes with pressurised ANFO loader.

The holes are primed with inverse initation system.

After charging is over the end of the hole is plugged with mixture of plastic clay and sand upto a depth around 90 cm.

(In modern practice after the bottom priming is over the blast holes are filled up completely with ANFO).

Total quantity of explosives per round of blasting

(20 blast hole) – 100 kg (20 kg 60% S.G. and 80 kg ANFO)

Volume of blasted ore = 5 m x 4 m x 4 m = 80 m^3

Pull per round – 4 m

If the denisity of ore – 3.2 tonne/m^3

Then tonnage of blasted ore/round of blasting = 80 x 3.2 = 256 tonne

Powder factor – 2.56 tonne/kg of explosive

Loading is done with LHD (capacity of the bucket - 2.3 m^3) within 4 hours of time.

A face OMS of 20 to 40 may be achieved with this type of method.

Hydraulic sand filling (70% to 75% solids by weight) is done at the rate of 200 to 250 tonne/hour.

Question 185 : Discuss drilling and blasting practices of the most modern highly mechanised SUB-LEVEL CAVING method of stoping with ring drilling and blasting.

Answer : A schematic layout of the most modern highly mechanised SUB-LEVEL CAVING method with ring drilling and blasting has been shown in the Fig. 91. The thickness of the ore body is around 20 m and

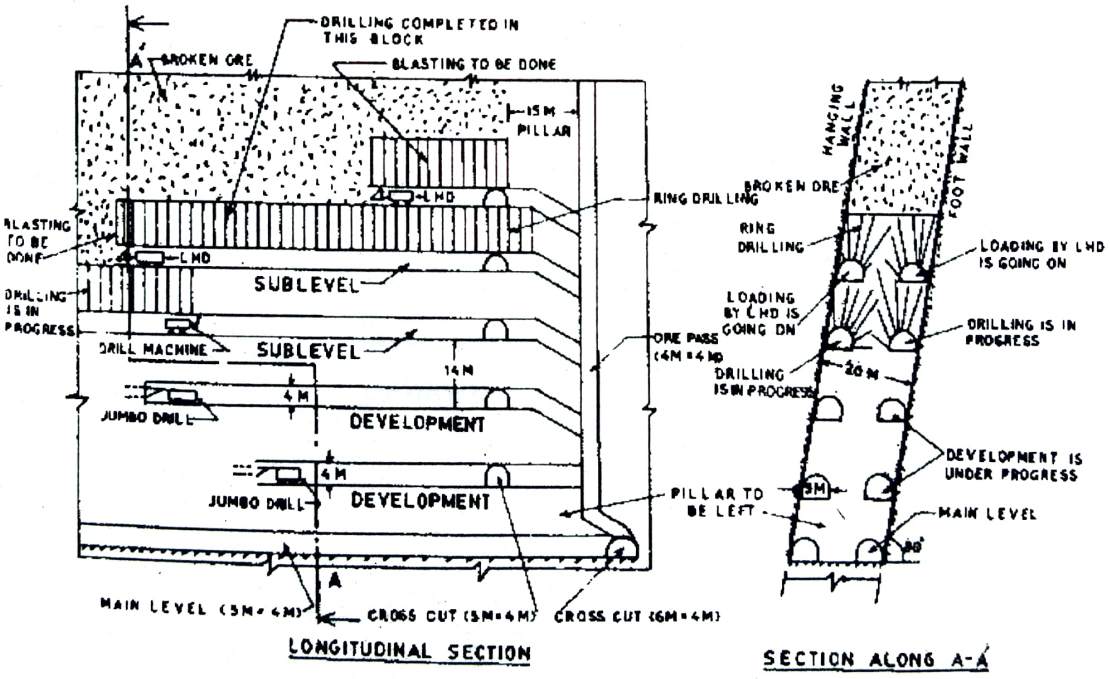


Fig. 91 Shows a layout of the modern sublevel caving method along with ring drilling and blasting practices (not to scale)

dipping at 80^0, occured in the hilly terrain. Opening of the deposit is done by the adits. Strike length - 2 km. The distance between two sub-level is 14 m with a cross-section of each sublevel is 5 m in width and 4 m in height to facilitate the efficient use of remote controlled LHD, electro-hydraulic jumbo drill machine, etc. Sublevel is driven both in hanging-wall and footwall which are connected by the cross cut of cross-section 5 m in width and 4 m in height. The size of the main cross cut near the ore-pass (size 4 m x 4 m) to collect the blasted ore efficiently is 6 m in width and 4 m in height. A pillar of 15 m width is left in between the stope and the ore-pass.

Drilling in the stoping and in the development faces are done by the tyre mounted 2 boom Electro-Hydraulic Drill Jumbos. In the stope ring drilling and blastiag is practised. Orebody above one sub-level is generally completely ready with ring drilling in the pattern shown in the fig. 91, before finishing a block above another sub-level which is above the former said sub-level. Blocks above the sub-level are mined out in sequence from top to downward. Broken ore is cleaned off by the remote controlled LHD (capacity of the bucket is $2.3m^3$) into the ore pass.

Drilling in the stope

It is done by the tyre mounted electro-hydraulic drill jumbo from the sub-level drives at a pre-determined burden and spacing.

The dia of the blasthole - 70 mm

Bit used - Button bit (cross bit may also be used)

Toe Spacing	-	1.8 to 2 m
Ring Burden	-	1.5 to 1.8 m
Angle of holes	-	It is predetermined and varies according to the conditions.
Length of the holes	-	It varies according to the angle of the blastholes. For this case the length of the longest drill hole may be around 14 m.

Rings are drilled in staggered pattern to allow a 0.9 to 1 m toe spacing on each alternate ring in order to reduce the blast effect on incompetent walls.

Average drilling/day/drill jumbo – 120 to 130 m

Drilling is done by one driller with one helper per drill jumbo.

Life of the drill bit depends upon the hardness and mineralogy of the ore body.

Blasting

Ringholes are blasted in the manner one ring/firing.

ANFO is used here as the main charge along with Primex cast booster. Primex cast booster is attached with raydet to achieve delays between detonation of the ring blastholes. Milli second delays used in this case. Each hole is single primed and placed as close to the toe as possible. Holes are blocked off by running a 32 mm dia. polythene charging hose and at the end of it a valve is fitted completely to block the holes. The hose is connected to a pneumatic pressure loader. First of all ANFO is loaded into the hole at a length varying from 0.5 to 1 m depending upon the length of the hole by the pneumatic pressure loader (Fig. 92). Then the primer (50 mm dia) is inserted into the hole with the help of the same polythene charging hose. Next ANFO is loaded into the hole up to the predetermined length. Collar of the hole is plugged (1 m to 1.5 m length depending upon the length of the hole) with the mixture of material like plastic clay and sand.

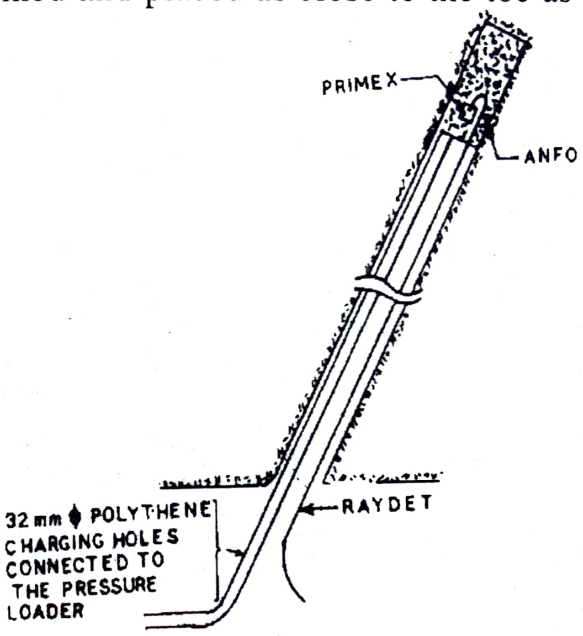


Fig. 92 Shows the details about the charging into the ring drilling hole (70m)

Amount of ANFO in the hole/m length – 3.4 kg/m length

Pull per ring – 1.5 to 1.8 m (say average – 1.6 m)

Average production per ring = 1.6 m x 20 x 10 = 320 m^3

If the density of the ore is 3.2 tonne/m^3 then production per ring = 320 x 3.2 = 1024 tonne.

Generally powder factor obtained by ring drilling and blasting in the sub-level caing method : 1.9 to 2.5 tonne/kg of explosive. A 2 tonne capacity pneumatic pressurised ANFO loader is sufficient for this case.

Question 186 : Discuss elaborately the drilling and blasting practices in a

modern LONGHOLE STOPING in Retreat method.

Answer : A typical modern longhole stoping layouts with longhole drilling and blasting method (vertical crater retreat mining method) shown in the Fig. 93 in the ore body of thickness 5 m dipping at 85^0 occured in hilly terrain. Opening of the deposit is done by the adits. Extent of the ore body along strike – 1.5 km. With this bulk mining system less amount of development work in the extraction block (no need of any development work for drawing of ore), and higher productivity is achieved. It will also save overall cost in drilling and blasting since less amount of drilling is required in this case. Only diameter of holes, spacing and burden increased. To control dilution; overbreak into the stopping walls and ore losses in the ore body, the deposit is blocked by driving levels (5 m wide and 5 m height) at an interval of 30 m and the ore block in between the two levels are further blocked by pillars (5 m width) placed at an interval of 20 m along the strike length. Cable bolting in the walls and in the stopes, controlled blasting, rapid filling of the exhausted stope (Goaf) by deslimed mill tailings/sand filling, etc. practised in this system of mining. Production starts, from the bottom most block advancing towards the strike length and after completely taking out of ore from each block they are filled with deslimed mill tailings/sand filling at the rate of 200 to 250 tonne/hour; through a hole of 400 mm diameter. Mucking is done from the bottom drive by a remote controlled diesel LHD (bucket capacity – 2.3 m^3).

Drilling

First of all a raise (2.4 m x 2.4 m) is to be made in the each block in the manner as mentioned in the Question No. 181. Then this raise is to be widened across the ore body (from footwall to hanging wall) by longhole drilling and blasting to make a slot cutting which will act as a free face for the remaining rows of ring blastholes in the stope. The blastholes are drilled by crawler drill models ROC-306 along with booster compressor (Model DG-418) pack of Atlas Copco.

Blastholes in each ring are drilled parallel to the footwalls and hanging walls to avoid the wall damages and blastholes are precollared with PVC collar pipe for preventing collapsing of holes. Inflatted rubber bladder should be used to cover up the collar of the holes just after finishing the drilling to prevent blockage in the holes by any foreign materials.

Length of the hole	-	25 m
Diametar of hole	-	115 mm
Burden of holes	-	2.5 m
Spacing of holes	-	2.4 m

A drilling rate of 30 to 40 m per shift is possible provided the supplied pressure by the portable booster compressor is very good. The entire stope is to be drilled first before the commencement of blasting to avoid any congestion and complicacy.

Blasting

After making raise and slots, ringholes are to be blasted in the manner one ring/firing. To guard against the wall damages and to reduce ground vibration the holes are charged in decks with ANFO to reduce the amount of explosives (not more than 90 kg explosive per deck is to be used). Pentolite cast booster along with 'Raydet' is effectively used to achieve delays between detonation of decks. Millisecond delays used in this case (Fig. 93).

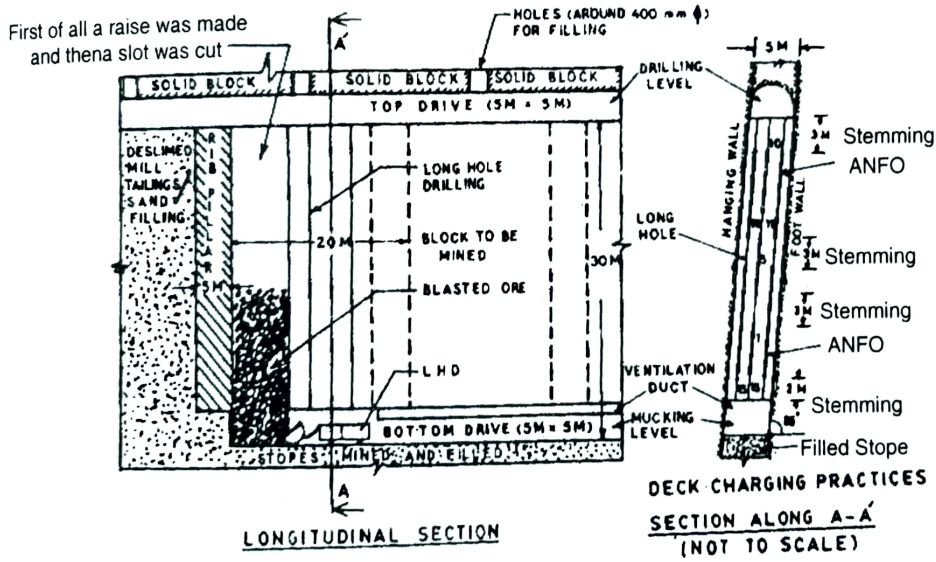


Fig. 93 Shows a typical modern longhole stoping layouts with drilling and blasting practices

During charging, the bottom of the holes should be plugged and then shall be stemmed with sand upto a height of 2 m. Then ANFO shall be poured around a height of 1 m above the stemming and pentolite cast booster along with Raydet which will act as primer shall be placed over it. Next ANFO will be poured over the primer upto the predetermined deck height. In between the two decks a 3 m space should be filled with

sand stemming and also at the end stemming is to be done upto a height of 3 m.

Amount of ANFO in the each of two side holes – 155 kg (approx.)

Amount of ANFO in the middle hole – 130 kg (approx.)

So total charge/ring = 2 x 155 + 130 = 440 kg

Pull per ring = 2.6 m (approx.)

Average production/ring = 2.6 m x 25 m x 5 m = 325 m^3

If the density of the ore is 3.2 tonne/m^3

the production/ring = 325 x 3.2 = 1040 tonne

$$\text{Powder factor} = \frac{1040}{440} = 2.36 \text{ tonne/kg of explosive}$$

For high speed loading, higher loading density, improved borehole coupling, productivity, safety etc. ANFO is loaded by pneumatic pressure by a pressurised ANFO loader fitted with a charge up truck. The capacity of the pressurised ANFO loader varies according to the need (it may be in any range between 0.225 tonne to over 10 tonne capacity).

The faces may be ventilated with two stage auxiliary fans of capacity 7.1 m^3/sec.

In the present system of longhole drilling and blasting generally deck charging is eliminated. The holes (since they punctured the bottom drive) are plugged at the bottom (at a distance of 0.5 m from the bottom edge) by polythene bags which are expanded by blasting a small explosive and block the hole. Stemming above the hole (1 to 1.5 m) is done either with moist sand or by moist drill cuttings. Before inserting the primer 3 to 4 m length of the holes are to be filled with explosives (ANFO, etc.) so that the position of the primer is above

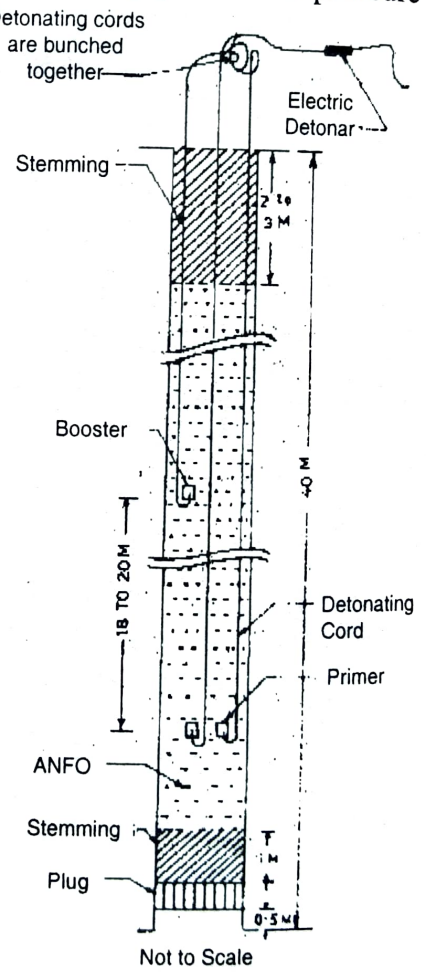


Fig. 94 Shows a details about the longhole blasting practice

the burden distance of that sized hole. Then pour ANFO with the pneumatic pressurised ANFO loader with the help of 40 to 45 mm dia. polythene charging hose. In the larger dia. holes (more than 115 mm dia) two primer made of cast booster are used in the column of charge. If the length of the hole is more, then another booster is to be used at an interval of 18 to 20 m from the primer. Detonating cords are bunched together at the surface and holes are fired by an electric millisecond delay detonator. Separate delays are to be used for each of the blasthole. Collar of the hole is plugged with the stemming material at a length of 2 to 3 m (Fig. 94).

Question 187 : Briefly discuss about various loading arrangements of explosives in the underground mines.

Answer : **ANFO loading in the underground mines.**

In the underground metal mines, pneumatic loaders using standard mine air pressure is used to load ANFO into the holes.

These are of two types which are as following -

(i) The ejector type (ii) The pressure vessel type

The Ejector Type - It consists of a hopper, an ejector assembly and control valve. It operates based on the ventury system whereby the premixed ANFO mixture is drawn from the hopper into the centre of an annular air jet within the ejector assembly and is then ejected into the borehole. Loading rate vary from 3 to 10 kg/min and by varying the loading hose dimensions it is possible to vary the particle size and thus the density of the loaded ANFO from 0.9 to 1.0 gm/cm^3.

The pressure Vessel type - It consists of a pressure vessel, a regulator and a control panel. The ANFO is maintained at a pressure not more than 2.1 kg/cm^2 in the vessel. The ANFO is then pressurised via a control valve into the discharge hose which may be upto 60 m in length. A discharge rate around 40 kg/min is possible with this method. The whole unit may be truck mounted and very much flexible in operation for bulkhandling of ANFO for small dia holes in the underground metal mines.

Presently bulk SMS/SME pumps trucks having capacity ranging from 0.5 to 12 tonnes for ring blasting is also in use in many underground metal mines. It is already mentioned that site mixed slurry (SMS) or site mixed emulsion (SME) system consists basically of two parts which are-

(a) A support plant (b) Pump trucks.

(a) Support plant - It is located either very near to a mine or centrally located to serve a group of mines. Non explosive ingredients are stored in it. Certain intermediates are prepared from some of these ingredients and kept ready for the use of mine.

(b) Pump trucks – These are specially designed to carry all the ingredients and pump the slurry into bore holes through a delivery hose carried on the truck which is controlled by the sophisticated controlling system for pumping operation of explosive to a predetermined quantity. This system provides various advantages which are as following :

- provides higher loading rate and density,
- provides versatility to charge explosive energy into the boreholes with better coupling which saves huge amount of cost,
- provides flexibility to change the density of explosives and capable to change three or more explosives formulations in predetermined quantities in the blastholes,
- improve both productivity and safety,
- eliminates the need for extensive magazine storage areas in the underground mining system.

Cartridge loading

For loading cartridge explosives pneumatic cartridge loaders are used to load the former into long boreholes. In ring drilling and blasting operation upholes are suitably charged with the use of these loaders with good charging capacity and high charging density. These loaders are compressed air operated and use antistatic materials to avoid incendive sparking.

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Chapter 5

Theory of Rock Fragmentation by Blasting

Question 188 : Discuss elaborately about the mechanism of fragmentation of rock by blasting including the thermodynamic aspects.

Answer : It is already deiscussed that the fragmentation of rock depends upon various parameters like explosive parameters, geo-technical parameters, initiation system, various controlling measures taken, blast geometry, etc. Since design of blasting parameters depend upon the site specific condition, all the theoretical approaches to explain the mechanism of fragmentation of rocks are empirical. Rocks fail on blasting operation due to crushing, tensile fracture, breaking due to reflection, release of load, generation of strain energy and extension of natural cracks due to expansion of gases, shearing action, inflight collision, flexural repture, etc.

When a piece of explosive is initiated then the site just around the drill hole will be crushed and plastically deformed. Beyond that crushed zone tangential shock waves causing heavy strain and stresses will be propagated at a velocity of 2000 to more than 8000m/s. These tangential stresses are responsible for extension of the cracks beyond the crushed zone. The shock wave so produced is first appears mainly in the form of compression which is then converted into tensile stress.

The stress waves propagates and attenuates with distance. The gases produced by the explosive, pressurise the rock around the wall of the hole causing growth of the cavity. The gases are surrounded by the quasistatic stress field in the rock. The hoop stress produced by the explosion of the charge is tensile in nature. It initiates the micro-cracks formed by the initial stress to continue to grow and connect them. The cracks run in planes normal to the direction of maximum tensile stress. Cracks starts at some distance radially from the cavity and when they grow up and finally penetrate the cavity, gases slowly inserted into the cracks producing additional faces and elongate the cracks. At the time of expansion of cavity and radial growth of the cracks, the stress wave

touches the free face and reflect back as rarefaction (Fig. 95 and 96). If

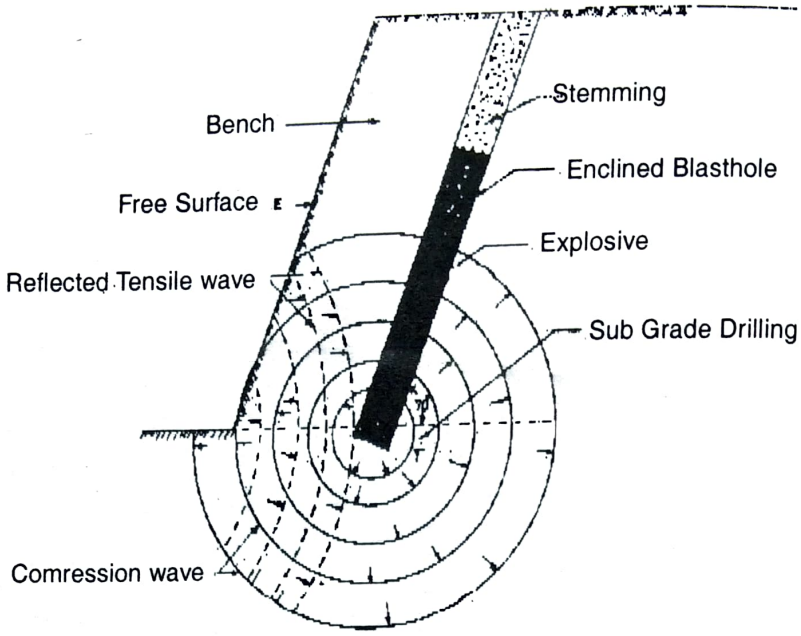

Fig. 95

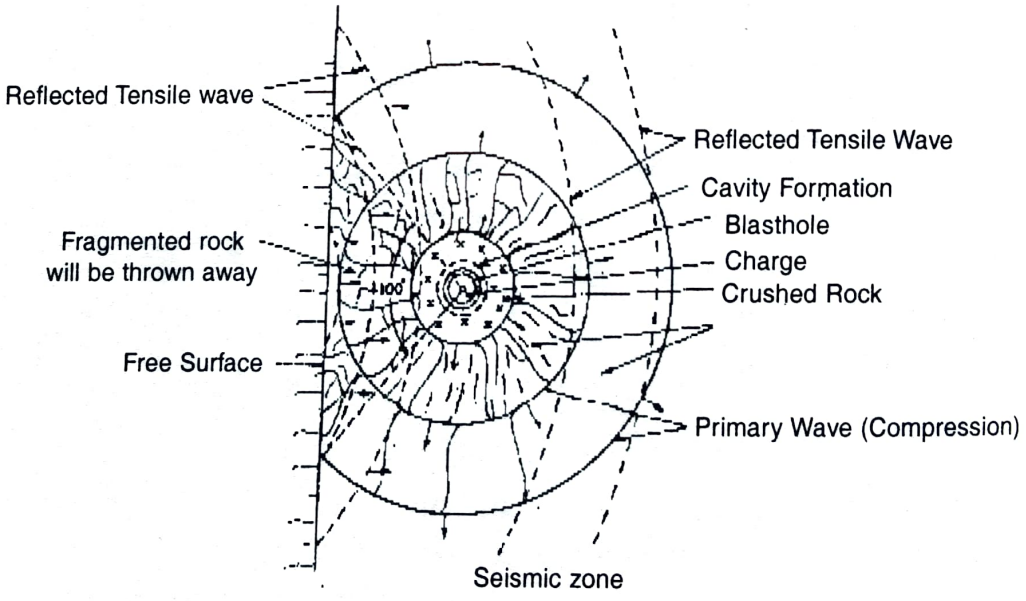

Fig. 96

the strength of the rarefactive wave exceeds the tensile strength of the rock, face spalling will occur. However, the weak rarefaction tensile stress wave causes the extension and growth of the cracks towards the free surface. As the rarefaction wave continues and partially unloads the cavity, fractured rock is accelerated and is thrown away from the surface. Rock near to the cavity receives additional acceleration from the expanding gases. The stress energy in the front of shockwave (at a

distance nearly twice the diameter of the hole) is greater than the compression. So it is again worth mention that crushing of rock occurs in a very narrow zone and in a very little amount of rock mass near to the explosive charge which causes a very high degree of dissipation of strain wave energy but do a very little job for the fragmentation of rock. Hence it would be better if the pressure in the blasthole is kept less than the crushing strength of the rock by good coupling. In and around the crushed zone radial fractures are developed due to the tensile strain accompanied by the high radial compression at the front of the elastic strain wave. A huge amount of rock fragmentation is affected by this radial tensile failure all around the blasthole irrespective of free faces. The peak strain occur at the wall of the blasthole and the lengths of cracks increase linearly. When the stored blast energy suddenly gets released in the manner of rock burst in the direction of the blasthole then it will cause setting up tensile stresses and disintegrate huge amount of burden. Pressure drops down suddenly as the gases escape through the radial fractures of load commences from the blasthole point and moves towards the free faces. Breaking due to internal reflection at the open joints and extension of suitable radial cracks are caused by the reflected tensile waves in a large burdens with or without causing any spalling. If the blast holes are stemmed efficiently then the gas pressure will produce the strain energy and will influence the extent of radial fracturing which assists good fragmentation of the rock mass by the explosives. However, this effect will be reduced if the gas escapes due to the wrong design of blasting parameters like small burden, less stemming in the blastholes, etc.

Shearing failure occures due to the relative movement of the rock burden along the suitably oriented fractures, or structurally weak planes which is caused by the heaving effect of the high pressure gases produced by the initiation of the explosives in the blastholes. The segments formed by the radial tensile cracks generates high tensile stress along their length causes transverse fracturing and flexural rupture. Secondary fracturing of rocks are also caused due to the inflight collision of the fragmented rock mass, which is generally influenced by the delay intervals and initiation sequence. If the total explosive energy is divided into two parts viz. shock wave energy and gas pressure energy, then the shock wave energy which is associated with the compressive strain is responsible for fracturing of rock by crushing, radial cracks, release of load, reflection

breaking of rock, etc. The pressure energy is mainly responsible for shearing fracture, flexural rupture, displacement of rock, extension of natural cracks and induced cracks by the strain wave, etc. Both the shock wave energy and the gas pressure energy depend upon the composition of the explosive and the V.O.D. (velocity of detonation). More the V.O.D. more would be the shock wave energy and less would be the gas pressure energy. It is worth mention that the shock wave energy in case of a high explosives accounts only 5 to 12% of the total energy and 70 to 80% of the shock wave energy wasted without doing any effective work. The breakage angle of any blasthole varies from 90^0 to 130^0 depending upon the various blasting parameters. The gas pressure energy expands the radial tensile cracks due to which the rock mass along the free face yields and moves for disintegration. In case of instantaneous blasting the gas pressure energy of all the holes assist together to move the rock mass along the free face for disintegration. If the spacing is increased instead of burden distance then the shearing and the strain energy will be increased.

In any blasting operation only 5 to 12% of the total explosive energy is used for fragmentation, displacement and throw of the rock and out of which only 50% is used in creating new surface area. Hence 1% increase in the use of explosive energy is a very big thing. It is already mentioned that the mixture of compounds of explosive undergoes a rapid self propagating extothermic reaction induced by either heat or impact or shock. The reaction when moves through the explosives at a faster rate in the unreacted explosive it is called a detonation and the velocity at which the reaction front moves forward through the charge is called the detonation velocity. It's range varies from 2000 to more than 8000 m/s. Faster velocity produces strong shock wave which causes the shattering effect. When the reaction is slower in the unreacted explosive, it is called the deflagration which causes mainly the heaving effect and very less shattering effect. The explosion is a rapid oxidation reaction in which all elements necessary for the reaction are present in the explosive body and the zone of reaction is very definite but its width varies according to the coarseness of the explosive ingredients. This reaction zone varies from a fraction of millimeter in case of fine grained high explosive to several centimetres in case of bulk explosives. The reaction produces various gases which exert tremendous pressure during expansion. The detonation

pressure i.e. the pressure of the detonation wave propagates through the explosives, is measured from the C. J. plane. The borehole pressure (ranging from 10000 to more than 60000 kgf/cm^2) which is exerted on the borehole wall mainly causes displacement of rock and also breaking. The explosion is adiabatic in nature and forms the thermodynamic equations. The temperature produced by an explosive after explosion is more than 5500^0c. The energy yielded by a gelatinous explosive may vary from 650 to more than 1400 Kcal/kg of explosive whereas oridnary ANFO and aluminized mixture of Ammonium Nitrate yield energy nearly 900 Kcal/kg and 1450 Kcal/kg of explosives respectively. Slurry explosives generate high persistent gas pressure.

Question 189 : Write short notes on the following :

(a) Shock energy of an explosive.

(b) Heave energy of an explosive.

Answer :

Shock energy (SE) : This energy accompanies the rapid change of state from a solid explosive (or liquid explosive) to a gas which occurs at the detonation front of the explosive. The fracture potential of an explosive is directly related to the reaction of explosive in the detonation zone. The shock energy is evaluated in the under water tests and the following equation may be valid :

$$SE \propto R^2 \int [P(t)]^2 dt$$

Where R = distance of measuring point (by a pressure guage) in the water (lake) from the source of explosion (by a sample explosive)

P(t) =pressure-time factor measured by the pressure gauge.

The peak blasthole strain (ε_P), and strain (ε) developed by an explosive in a blasthole can be expressed as follows :

$$\varepsilon_P = \frac{P_{eQ}(1-\gamma)}{2E}$$

$$\varepsilon = \frac{PPV}{V_P}$$

Where P_{eQ} = final equilibrium pressure of explosion gas.

γ = poisson's ratio.

E = Young's modulus for the rock.

PPV = pick particle velocity

Vp = rock compressional wave velocity.

The level of induced strain, vibration and shock energy depends mainly upon the burden, spacing, geo-mechanical properties of rock, density of rock, bulk modulus, degree of saturation of water explosive properties and the performance. More the PPV, more would be the rock fragmentation potential of the explosive. The value of PPV obtained at a fixed distance behind the blastholes charged with equal amounts of different explosives in the same kind of rock with identical burden and spacing.

Shock energy is responsible for fragmentation whereas heave energy is mainly responsible for muckpile displacement. These two energies after blasting are strongly influenced by confinement and charge diameter. The VOD is directly related to the detonation pressure generated in the blasthole and it can be simply represented by the following equation :

$$Dp \propto \partial \exp \times VDO^{2}\text{actual}$$

and $$n = \left(\frac{VOD_{actual}}{VOD_{max}} \right)^{2}$$

where Dp = detonation pressure produced by the explosive in the blasthole, Pa

∂_{exp} = density of explosive, Kg/m^3

VOD_{actual} = actual velocity of detonation in the blasthole, m/s.

VOD_{max} = maximum VOD of explosive in steady state, m/s

n = Volumn fraction of explosive contributing to the detonation reaction which produces the shock energy. (The remainder of explosive reacts behind the detonation front produce heave energy).

Heave energy : Heave energy of an explosive is it's ability to displace the rock burden and it is due to peak blasthole pressure produced in the blasthole in the region behind the primary reaction zone.

The rate and timing of burden movement by this pressure depend on the size of burden and density of rock or the total mass of rocks displaced. The kinetic energy of the burden (KE_b) is expressed by'

$$KE_b = \frac{mV^2}{2}$$

where m = mass of rock moved (Kg)

V = average velocity of movement of rockmass, m/s.

The burden velocity can be measured using high speed camera or video recorders, the latter in conjunction with computer digitization. However average burden velocity can frequently be calculated by considering the motion of centre of gravity of the rock-mass (burden). After locating the centre of gravity of the pre-blast and postblast rockmass (burden), the average velocity of movement (Vave) for the entire muckpile is determined by the following equation :

$$V_{ave} = X\left(\frac{g}{2y}\right)^{0.5}$$

where X = displacement of the centre of gravity in the horizontal, m

Y = displacement of the centre of gravity in the vertical, m

g = acceleration due to gravity in m/s^2.

When two explosives charges are simultaneously initiated with an air gap in the centre, the shock waves from the two explosive charges collide together, causing a secondary breakage effect in the centre where there was no explosive. Much less energy is wasted in pulverising the rock in the crushed zone and a greater amount of energy is available for useful work. Thus using the air-decking technique the following results may be obtained :

- uniform and improved fragmentation, controlled throw, reduction of back-breaks and fines.
- higher use of explosive energy for the useful work.
- saving of explosives may be ranging from 10% to 40%. On an average the savings of explosives may be 15% to 20%.
- the air decked holes can withstand sleeping of several days.

As per Berter (1990) the energy transmitted to the rock may be distributed in the following manner :

i)	Fracture in-situ	< 1%
ii)	Breakage	15%
iii)	Displacement	4%
iv)	Crushing in the vicinity of the hole	1.5 - 2.0%
v)	Fly rock	< 1%
vi)	Deformation of solid rock beyond the shot	< 1%
vii)	Ground vibration	40%
viii)	Air over pressure	38-39%

Question 190 : Discuss parameters of Rock Breakage, Burden, Spacing, Good Fragmentation, Blasthole Diameter, Subgrade Drilling, Stemming initiation, etc.

Answer : **Parameters of Rock Breakage**

The rock breakage parameters are broadly classified into the following :

Explosive properties, Rock properties; Blast Geometry and Charge loading properties; Safety and specific problems.

Explosive properties – The main properties under this heading are V.O.D. detonation pressure and impedance, energy factor and density of explosive, gas pressure and shock wave energy, volume of gas, composition of explosives, type, nature and strength of explosives, type of detonation, primers, powder factor, etc.

Rock Properties – The main properties under this heading are the dip, strike, compressive strength, tensile strength, shear strength, density, elastic property, bedding planes, structure, etc. of rocks, presence of geological disturbances like faults, folds, fractured ground, etc. joints, cleats, cleavage, etc, specific damping capacity, cohesive resistance, heterogeneity of rock etc. Beside these the strain wave attenuation, propagation velocity of the shock wave, characteristic impedence, etc. are also some of the important properties of the rocks. If the specific damping capacity decreases, the shock wave energy and strain generated fracturing will increase and vice-versa.

Charge loading parameters and Blast geometry – In this category the parameters which are included are diameter and the length of

shotholes and charges, stemming materials and height of stemming, degree of decoupling, method and sequence of initiation, blasthole diameter, spacing and burden, distribution of explosives along the hole, loading density, angle of blasthole, number of holes in a row and number of rows, subgrade drilling, local restriction due to the proximity of important installation or built-up areas, watery and climatic condition, amount of strata to be broken, requirement of shape of the excavation, factors of loading, transporting, and requirement of crushing and screening, working area, casting, etc.

Safety and specific problems – The safety hazards and problems which are associated under this heading are air blast, noise, vibration, proximity of dwellings, throwage, slope instability, back break, oversize, toe and floor problem, caving of hole, overhange, cleaning, etc.

Burden and Spacing – If burden (distance between the bottom of the hole and the free face) is less than the optimum size then fracturing by strain energy increases and that by the gas pressure energy decreases since the strain energy is so high that the gas pressure energy is wasted at the atmosphere. But when the burden is more than the optimum value, the strain wave can not efficiently fracture the huge amount of burden mass and also the gas pressure energy can not be efficiently utilised. Hence optimum burden (which depends upon the site specific condition) is necessary for good degree of fragmentation of high volume of rock mass. Spacing (Distance between the two blastholes in the same row) is usually selected between 1 to 2 times the burden thickness depending upon the parameters of rock. The spacing influences the nature of the face to be created behind and degree of fragmentation. The spacing may be designed 3 to more than 7 times greater than the burden for optimum fragmentation of rock which is in contrast to the conventional spacing to burden ratios. A short delay single row blast is more productive than the instantaneous single-row blast as larger spacing can be employed in this case for the same burden. Staggered configuration of initiation with delay times enhances the fragmentation and facilitates to increase the spacing of holes. The size of burden depends upon the charge per metre at the bottom of each hole, the charge height, spacing of the holes, strength of the explosive, inclination of hole, rock parameters, etc. For obtaining a very good production, it is better if the spacing distance is increased instead of increasing the burden distance, otherwise there may be the chances of toe problem.

Good fragmentation, Blasthole diameter, etc.

A good fragmentation is designed in most of the cases but the degree of fragmentation requirement depends upon the systems like loading, transporting, crushing, screening, etc. A good fragmentation is obtained by making shorter spacing, burden and length of the holes, using short delay detonators, and using explosives which produce larger amount of gases and also by distributing charges evenly whole along the holes. A good fragmentation reduces the cost of secondary blasting.

During selection of the type and quantity of the explosives to be used the powder factor, safety, applicability to a particular case, total cost of operation e.g. cost in drilling, blasting, loading, crushing, screening etc., overall economy, are very important besides the explosive parameters, rock parameters, blast geometry and charge loading parameters which are already discussed. Blasthole diameter also depends upon the above mentioned factors. Recent trends are towards the use of larger diameter blastholes to reduce the cost of blasting. Larger dia. blastholes also enhance shock wave energy and tend towards the ideal detonation. But the larger dia. blastholes produce coarser fragmentation and mainly should be used for heavy blasting. Charge distribution in the smaller dia blastholes are better. The charge length contributes degree of radial fracturing, limit the spacing between the simultaneously initiated holes and influences flexural rupture. However, if the blasthole diameters are quite large then heavy boulders may be formed which will require intense secondary blasting. Hence at that time the diameter of the blastholes shall be made optimum by considering the various parameters of rock breakage. Low density, low cost explosives can efficiently be used in larger dia. blastholes whereas smaller dia. blastholes shall be employed in harder rock formation using higher density explosives.

All the cylindrical shape explosives have a characteristic critical diameter below which they will not function (will not propagate a detonation wave). The smaller the diameter of an explosive column, the smaller would be the efficiency of an explosive and also the VOD will lower down accordingly. However, the diameter plays a very important role in slurries and ANFO compare to the gelatinous explosives. The critical diameter of lead oxide and PETN are very small. Bulk explosives are generally designed to be used in the larger diameter boreholes. The critical diameter of ANFO is around 48 mm. If those explosives are to be

used lower than their critical diameter then their ingredients are to be made finer in size and costly ingredients must be added to make them function efficiently and explosives will become costlier. Smaller diameter drillholes need much closer spacing, burden, and require more number of holes to be used to blast in a particular volume of rock mass. The smaller diameter blastholes also require efficient loading equipment for slurry explosives and the rate of loading is also slower. The cost of blasting accessories, cost of drilling, loading and stemming are also very high in this case. The explosive diameter above the critical diameter provides efficient detonation and greater energy per unit weight of explosives. The critical diameter of an explosive depends upon the particle size, reactivity, density and confinement.

Subgrade Drilling, Stemming, Initiation

Subgrade drilling in most of the cases in surface mining operations are essential to negotiate thc toe problem for efficient loading operation, although when there is the presence of distinct bedding plane or joints near to the floor level then sub-grade drilling may be avoided. If the burden and spacing is increased then the sub-grade drilling has to be increased accordingly. Generally the sub-grade drilling length is 0.3 times the distance of the burden but if there is the presence of huge problem of toe then sub-grade drilling = 0.5 x Burden distance.

Bottom charge shall be concentrated and its length shall always be less than the length of the burden.

Although the stemming does not assist significantly for the effectiveness of strain wave energy but it reduces the premature venting of high pressure gases into the open atmosphere and help to fragment the rock mass due to gas pressure energy. A well stemming also assist to enhance the burden distance. The peak strain at the stemming level should be high for proper fragmentation at the collar zone of the hole. Relatively long stemming should be used if the mass of rock body has a higher degree of natural cracks and planes of weakness.

In a hole if two primers are used one shall be placed at the bottom of the blasthole and the other one located at such a point that the strain wave meets at the grade level. When the point of initiation is at least 0.25 time the burden below the stemming – charge interface then the generation of peak strain due to the interaction of the adjacent hole may be the maximum. In hard, brittle and massive rock the use of multiple

primer will enhance the peak strain at a given point in the burden. In delay firing of a single row blasting the gas pressure energy conserved by the reduction in throw will be used in the vertical shearing of the burden. When the delay between the adjacent holes is greater than 0.5 second then there will be no interaction or superimposition of the stress waves, but when the delay is in between 5 to 100 ms then very complex interaction of strain waves and in-flight collision of blasting fragments will occur. Depending upon the blasting parameters the time between detonation and the beginning of movement of the fragmented rock is usually 5 to 75 ms.

Question 191 : Write short notes on :

1) Livingstone crater Theory
2) Decoupling
3) C-J plane
4) Shaped charge

Answer :

(1) Livingstone crater Theory

If e = empirical strain energy factor which depends upon the geotechnical parameters of rock

M = Weight of the explosives just to fracture/make failure to a rock mass, in lbs

Dc = Critical depth of the charge in a hole, in ft.

Then $Dc = eM^{1/3}$ .....(1)

If D_0 = Optimum depth, in ft.

Δ = depth ratio (D_0/D_C)

Then modified formula of Livingstone Crater theory

$D_0 = \Delta e M^{1/3}$ .....(2)

Hence the optimum depth varies directly to the value e, M and Δ.

To determine the performance of an explosive three types of scaling in the crater investigation are indentified based on (a) energy (b) weight (c) volume. Explosives are generally compared on energy basis although volume scalling is considered in surface mining operation.

(2) Decoupling

It is the ratio of the diameter of the blasthole to the diameter of the

explosive. The amplitude of shock wave velocity in the rock is nearly inversely proportional to the 1.5 power of decoupling.

To bring down the peak blast hole pressure to a value near to the dynamic crushing strength of the rock decoupling is needed. The decoupling increases the effect of gas pressure energy at the expense of shock wave energy and improves fragmentation by 65-67%. Decoupling is very effective in well-stemmed, bottom primed charges in porous, plastic rocks with high specific damping capacity. But decoupling may not be effective in hard, brittle, massive rocks with low specific damping capacity which require more strain energy compare to the gas pressure energy. The degree of coupling is the amount/extent to which an explosive is in contact with the wall of blasthole. Loss of energy will be less if the coupling is good since loss of energy through the space in between the charge and the wall of the hole will be less in this case.

C-J Plane

It is the Chapman—Jouget plane. This is the plane in a column of charge placed in a blasthole where the reaction for the detonation shock wave is completed. It is considered from the rear plane of the reaction zone. The plane is also the starting point for the permanent steady state reaction of the onward explosive charges in the blasthole column. After the initiation of a high explosive column which is placed in a blasthole, a small amount of charge (primer) first initiated and the shock wave so produced will move towards the explosive column with a transient velocity to a particular distance (point) at a very small interval of time (in the order of fraction of millisecond) until the remaining charges are initiated by the high temperature and pressure of the produced gases due to the reaction at the detonation point. The detonation wave propagate at a steady state (velocity of detonation varying from 1500 m/s to 8000 m/s) beyond the C. J. plane and help to react with the remaining charge by exothermic reaction and complete the process of decomposition of the total explosives in the blasthole. Detonation pressure is measured from the C-J plane.

Shaped Charge

The basic components of a shape charge consists of a detonator, 1.5 kg high explosive, a cylindro conical body of 100mm dia. and a ceramic liner having an included angle nearly 50^0 to 55^0. The charged is placed over the conical liner about 175mm away from the rock face. When the

charge is initiated the liner is fragmented and ejected out as a jet with a velocity around 900 m/s and it makes a hole (around 75mm dia and 1.5m length) into the rock face with the principle of "Munroe Effect". The shaped charge technique can be very much useful in the under sea mining and also where it is very difficult to use conventional explosives. But the problems like effect of air blast, accidents due to flying projectiles are associated with this.

Question 192 : Discuss briefly about Blasting Efficiency in the underground mines.

Answer :

Blasting efficiency means an unit weight of explosive will yield maximum output producing minimum amount of fines and with high degree of safety.

There are many factors on which the blasting efficiency depends. There are many factors which are not under the control of human hand like strength of the mineral/rock, nature and types of rock/mineral, cleats and cleavage planes, dip and depth of the coal seam/mineral body, lithology, mineralogy, presence of faults, folds, dykes, other stone intrusions, watery condition, friability, etc.

But there are many factors which can be controlled by the man for the improvement of blasting efficiency which are as following —

In Headings/Development Faces

- Width and height of the gallery
- With cut by CCM, the depth and position (top, middle or bottom) of cuts in case of coal,
- Number and pattern of shotholes and their length (w.r.t. depth of cut by the CCM in case of coal)
- In solid blasting type of cut, number of shotholes and their length, width and height of the gallery are very important.
- Type and amount of explosives used,
- Nature of blasting e.g. whether blasting is done with delay detonators or not, whether single or multishot firing is used?
- Type and amount of stemming materials used in the shot holes and efficiency of the stemming materials, etc.

In Extraction Process

Here the following points are very important—

- Pattern of shotholes and their angles.
- Height of the mineral body/coal seam and method of stoping/ method of working in coal.
- In case of coal whether extraction process is by longwall method with or without any cut by CCM or splitting of gallery and extraction of stook in case of bord and pillar mining or by other methods of working are very important.
- Dimensions of stopes and strength of mineral body/rock also play very important roles during extraction process of mineral.
- Use of right type of explosives with optimum charge and optimum delay interval, proper spacing and burden of holes, optimum length and diameter of holes with proper inclination, etc. will definitely improve the blasting efficiency.

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Chapter 6

Other Stoping Practices

Question 193 : Enumerate the factors which govern the choice of the stoping methods.

Answer : The following are the main factors which govern the choice of the stoping methods : size of the mineral body (thin, moderately thick, thick), shape of the mineral body (bedded, lenses, loads, irregular shape), dip of the mineral body (flat to dipping below 37^0, from 37^0 to 50^0 and 51^0 and above), regular or irregular deposit, depth of the mineral body (shallow depth, medium depth, high depth), mineralogical characteristics of the mineral body, composition of mineral body (whether mineral occurs in native form like gold in the quartzite, number of metal contents in the mineral body like one, two, three or more, changes of composition of the same mineral body right from the surface to the higher depth, sulphide mineral body which catches fire due to spontaneous combustion and changes the characteristics of ore due to oxidation, contacts of mineral body with the country rocks, distribution of valuable constituents of mineral in the mineral body, etc.), strength and physiological characteristics of mineral body and the country rock (compressive strength, tensile strength, shear strength, plasticity, elasticity, young's modulus, drillability index, stability index of mineral body and the country rocks of hanging wall and foot wall, various stone or other intrusions in the mineral body, etc,); value of ore (contents of the various metals in the ore in percentage, the value of the right metal ingredients in the ore, dilution of ore during stoping operation, ete.), sizes of blasted ore (big boulders, larger sizes, smaller sizes, fines, etc. which depend upon mainly on drilling and blasting parameters and geotechnical parameters of rock), presence of surface structure (whether remote area, presence of villages or township or important buildings/industrial complex, etc.), watery condition, presence of old workings, location and sizes of old/ present shafts/inclinces/adits, availability of supporting material particularly timber in that area, amount of proved, inferred or indicated ore reserves in the mining area, availability of right type of indigenous

machinery for mechanisation of the stopes, availability of skilled manpower, stoping method which yields maximum profit with small investment, the method which extracts maximum percentage of ore from the mineral body with high production and productivity, stoping method which is the safest one, requirement of selective mining and grade control, presence of geological disturbances like faults, folds, and other tectonic disturbances, working environment, rate of production requirement, availability and types of filling materials (if cut and fill method of stoping is practised), availability of capital, cost economic criteria, etc.

OVERHAND STOPING

Question 194 : Discuss **Overhand Stoping** after enumerating its various advantages, disadvantages and applications.

Answer : In this case stoping is done from lower level to the upper level i.e. in ascending order and miners work below the stopes (Fig. 97). The

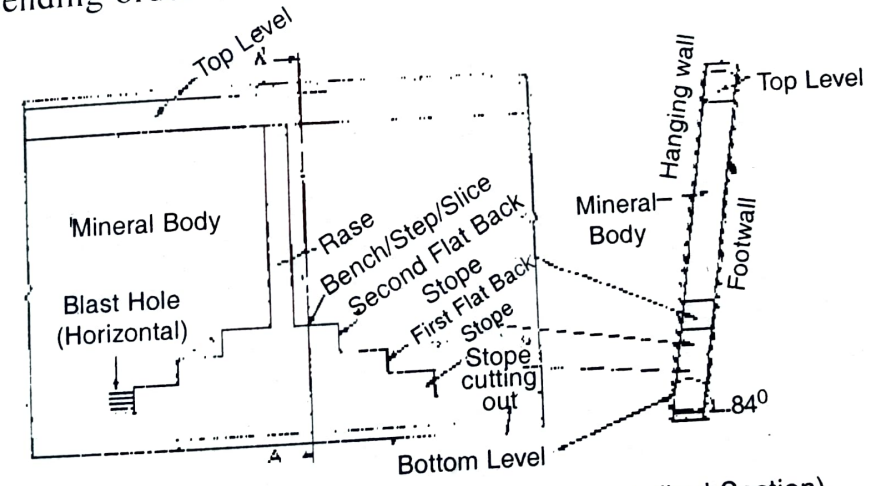


Fig. 97 Open Overhand Stoping Method (Longitudinal Section)

level interval may be in the range between 25 m to 70 m. If the ore body is very regular and blocky in nature then level differences are more, otherwise for irregular shape of the deposit the level differences are kept to minimum depending upon the site specific conditions. In the ore body first of all level drifts are driven (one or two in numbers in each level depending upon the thickness of ore body) which are then connected by raises kept at an interval of 40 m to 80 m depending upon the geotechnical parameters of mineral body, country rocks and also on site specific condition. In this way mineral blocks are formed. If the strata is

very strong and also the thickness of the mineral body is less than 2 m, then pillars are not left in the either side of the raises, but for weak mineral body and country rocks, pillars (4 to 15 m in thickness) are left in the either side of the raises. Sill pillars are left above the bottom level which has a thickness around 6 to 10 m and chute raises are driven through the sill pillars at an interval of 10 to 15 m (centre to centre) along the strike length. Generally during selection of pillars the low grade mineral body shall be selected. In place of sill pillars sometimes square set timbering may also be done for supporting the hanging wall. Stoping in the bench may be done either from one side or from both sides of the raises. Each lower horizontal slice (bench/step) is advanced by 8 to 15 m ahead of the upper horizontal slice (bench/step) in the strike direction and in chronological sequence (from bottom to top). Height of the each slice may be 1.5 to 3 m. If the roof and the floor conditions are very strong then no supporting is necessary in the stopes but otherwise stulls or square set timbering, roof bolting or cable bolting shall be done in the stopes or ore pillars may be left in the stope. Ore after blasting falls down by gravity if the dip of the vein/load is more than the angle of repose of the material (generally it should be more than 38^0). For lesser gradient, scrapers may be used for hauling ore into the loading chute. In steep gradient ladders shall be provided between the slices (benches/steps) and persons should work with safety belts over the temporary platform made by wooden poles/sleepers over the stulls. Horizontal slices (steps/benches) are generally practised in the open stopes where the hanging wall, footwall and mineral body are very strong but sometimes inclined slices are also practised in the cut and fill stoping. Air current passes from the lower haulage level to the upper level after ventilating the stopes properly. Holes are drilled either by the jack hammer or stoppers horizontally or vertically from bottom to top or in the inclined manner. Length of the holes varies depending upon the thickness of slices/benches/steps (say 1.5 to 3 m). Diameter of holes varies from 34 to 51 mm, burden and spacing of holes varies from 0.4 m to 1 m and 0.4 m to 1.2 m respectively. Average rate of drilling per stopper per shift is around 70 to 90 m.

Explosive used generally 60% Special Gelatine

Amount of explosive used per hole is around 350 gms to 490 gms (approx.) for a hole length of 1.5 m depending upon the hardness. For 51

mm dia. blast holes ANFO can be used with 60% S.G. as primer initiated with Nonel millisecond delay detonator and blasted with detonating cord.

UNDERHAND STOPING

Question 195 : Discuss **Underhand Stoping.**

Answer : In this case the stoping is done from upper level to the lower level i.e. in descending order and miners work aftenstanding over the mineral body (Fig.-98). The level interval may be in the range between

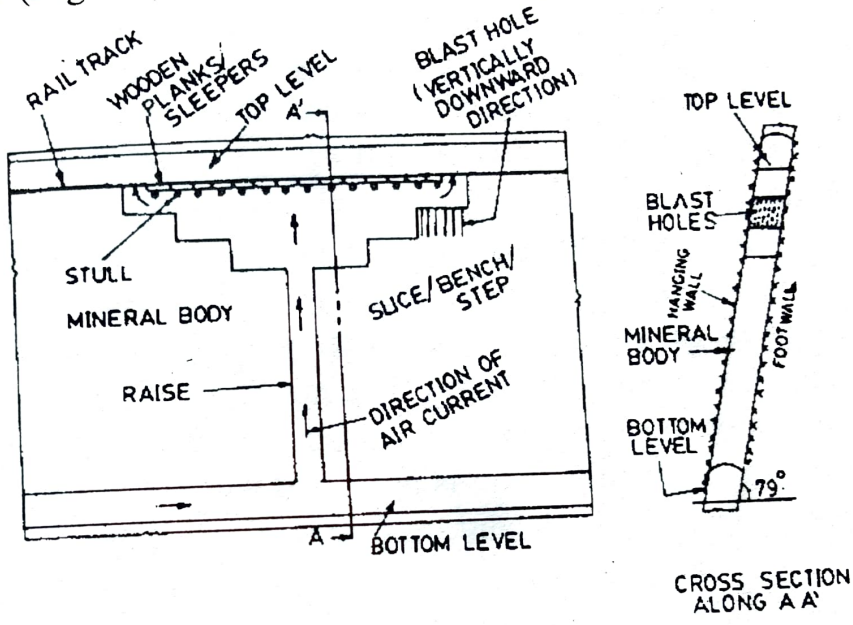


Fig. 98 Open Underhand Stoping Method (Longitudinal Section)

25 to 70 m depending upon the site specific conditions. If the ore body is very regular and blocky in nature then level differences are more otherwise the level differences are less in case of irregular shape of the deposit. Level drifts (either one or two in number) are driven by the raises/winzes, keeping at an interval of 40 to 80 m depending upon the site specific conditions and geotechnical parameters of the country rock or ore body and forms the blocks of ore. The mineral blocks are extracted in slices/benches/steps of which the upper slices/benches/steps are advanced ahead of the bottom forming converging towards the bottom of the stope. Stoping in the benches/steps may be done either from one side or from both the sides of the raises. The raises or winzes serve the purpose of the ore pass. The blasted ore is shovelled into the raises/ winzes which fall down by gravity and they are then loaded onto the mine cars in the bottom level. Afterward the mine cars are transported to

the required place by various haulage means like mine locomotive, rope haulages, etc. In this method the upper level is maintained either by leaving crown pillar of thickness varying from 5 to 8 m depending upon the strength of the mineral body when the grade of ore is poor but when the grade of ore is very rich then no pillar is left and in place of it stulls are erected and lagged properly and haulage tracks are laid over it. This underhand stoping is generally suitable for deposits having thickness varying from 1 to 3 m and dip varying from 45^0 and above in a very irregular deposit having varying grade of ore and also for small scale mining. This type of underhand stoping is also done where both the hanging and footwalls are very strong and also when the fault zone is to be negotiated. With this method, mining of the undesirable part of the mineral body (low grade ore) may be excluded, high percentage of mineral recovery and stoping of more number of blocks are possible. Ventilation in the stope is very easy but dilution of ore in the narrow ore body is quite large and also for supporting the stopes heavy amount of stulls or roof bolting are necessary. However, the dilution of ore in case of the steep veins are very less. Sometimes the stope opening is V shape and even this underhand stoping is practised without development of any raises (here winzes are made and progressively deepened) as the number of slices are increased from top to bottom, but cost of hoisting of blasted ore is more in this case. In wider veins when both the hanging and footwalls and also mineral body are strong then this method may be practised after leaving the ore pillars in the stoping area. In most of the cases level drifts are driven along the footwall and then they are connected by the raises. But for a very wider mineral body block, drivage of two drifts in the same level may be necessary which shall thereafter be connected with cross cuts. Above the main haulage level sill pillars of 6 to 10 m thick may be left and chute raises are driven in the sill pillars at an interval of 10 to 15 m (centre to centre along the strike length) and thereafter funnel is made in them connecting both the foot and hanging walls. This is done to collect the broken ore which is then loaded onto the mine cars/tubs, which are hauled afterwards with thc help of locomotives, rope haulages, etc. to the required destination.

Blastholes are generally drilled downward by the jack hammer or mounted hammer drills at a depth varying from 1.5 m to 3.5 m depending upon the thickness of the veins. Depending upon the depth of the hole, requirement of fragmentation, strength of the mineral body; the spacing

and burden varies from 0.5 m to 1.8 m and 0.4 m to 1 m respectively.

Explosive used generally 60% S.G. Amount of explosive used per hole is 350 gm to 490 gm (approx) for a hole length of 1.5 m in the mineral body. For 51 mm dia. blasthole ANFO can be used successfully with 60% S.G. as primer initiated with Nonel milli-second delay detonator and blasted with detonating cord.

Question 196 : Discuss briefly the Top Slicing.

Answer :

TOP SLICING

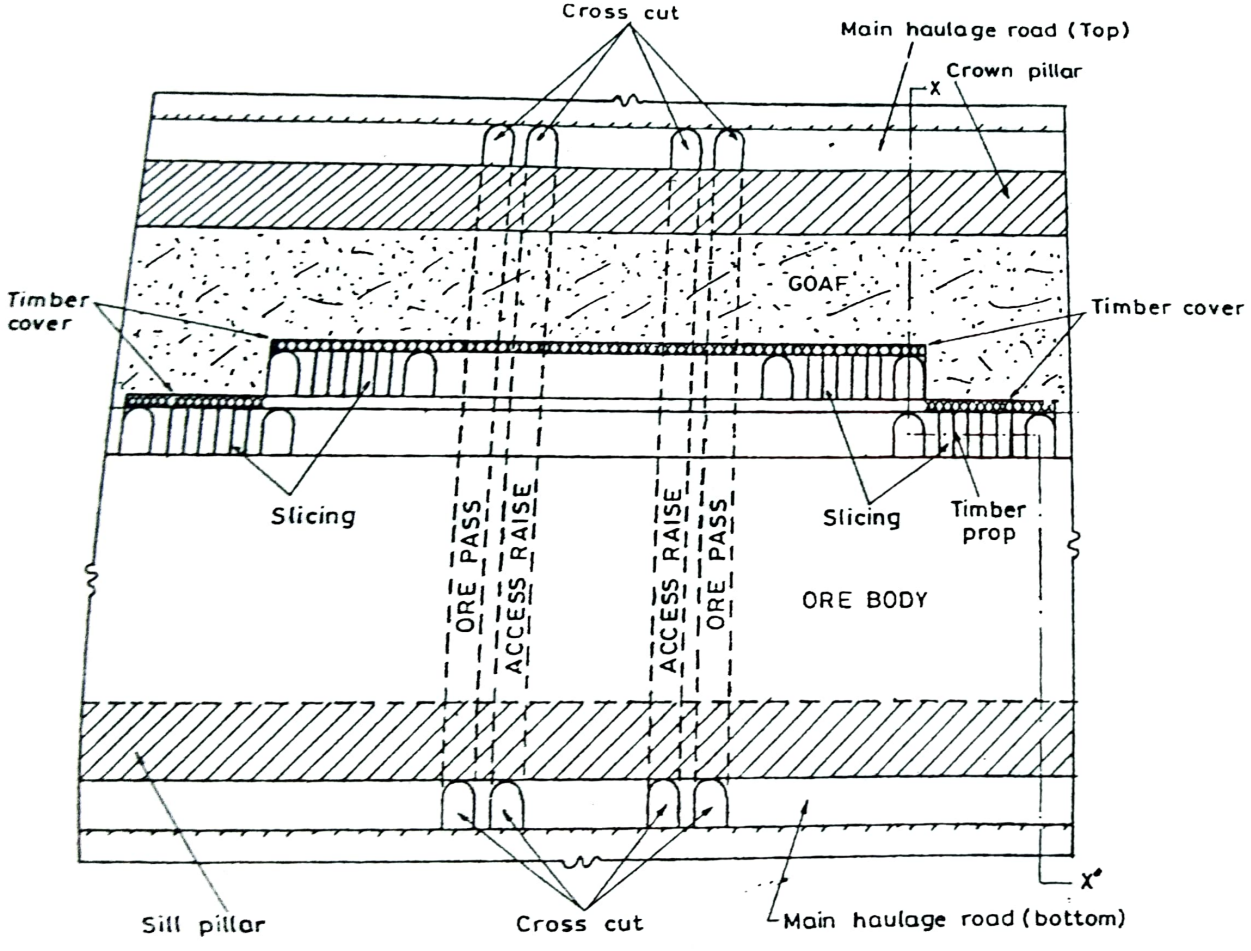
Top slicing method is generally suitable for thick and very large horizontal ore body. First of all raises shall be made in the ore body for the purpose of ventilation, manway, material transport and they shall be connected by the cross cut levels, keeping the distance between two cross cut around 40-50 m. Then the formed ore blocks in between these cross cuts shall be extracted from the either ends. Middle block may be extracted at the last.

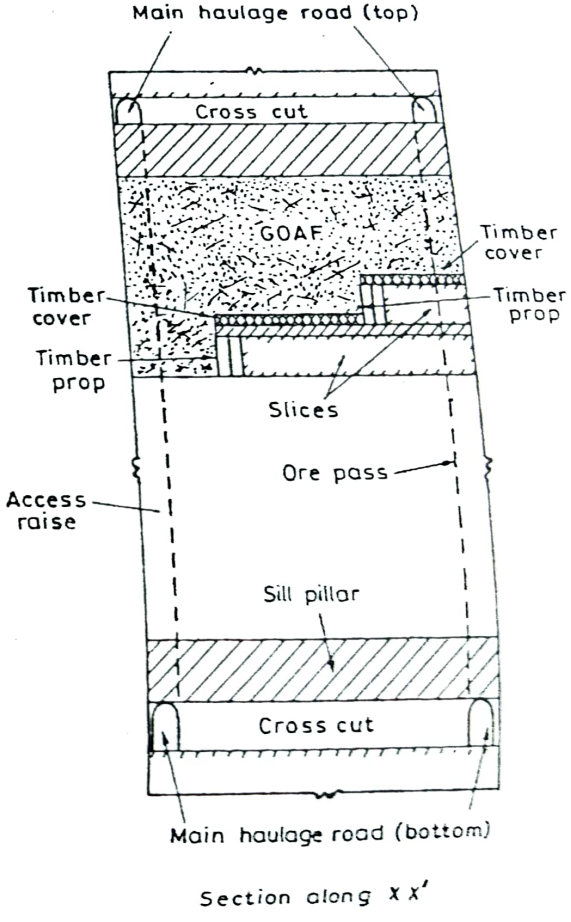
To extract each block parallel slices of the cross cut shall be driven and a mat of timber shall be laid on the floor of each drive which will act as an artificial roof in between the caved rockmass and the extracted ore of the next successive slices to support the face and also to control dilution of ore. After finishing the top slice in this manner the next bottom slice shall be extracted out under the caved rock mass. If necessary extra timber shall be laid down in each successive slices to construct more robust flexible roofing (Fig. 99).

Advantages of top slicing are many e.g. higher recovery rate, little dilution of ore, suitable for intermittent operations, operation is cheap (provided huge amount of timbers are availabe at the cheaper rate and the cost of labour should also be cheap), applicable under the weaker ground condition and also it is applicable where the practise of overhand stoping is not possible.

The demerits in this system are as follows :

Causes surface subsidence, availability of timber now a days is not only difficult but also very costly, fire hazards due to timber and sulphide ore, ventilation is difficult, troubles offered by the artificial roof, seggregation of high and low grade ore is difficult, gestation period is very long, huge requirement of manpower for handling of timber, etc.





Section along X X'

FIG – 99 TOP SLICING

STULLED STOPING AND SQUARE SET STOPING

Question 197 : Discuss briefly about the following :

A) Stulled Stoping

B) Square Set Stoping

Answer :

(A) STULLED STOPING

This is either An underhand or overhand method of stoping in which the stope faces are supported by the stulls. In overhand method of stoping the cut and fill method of stoping may be practised (fig. 100) if

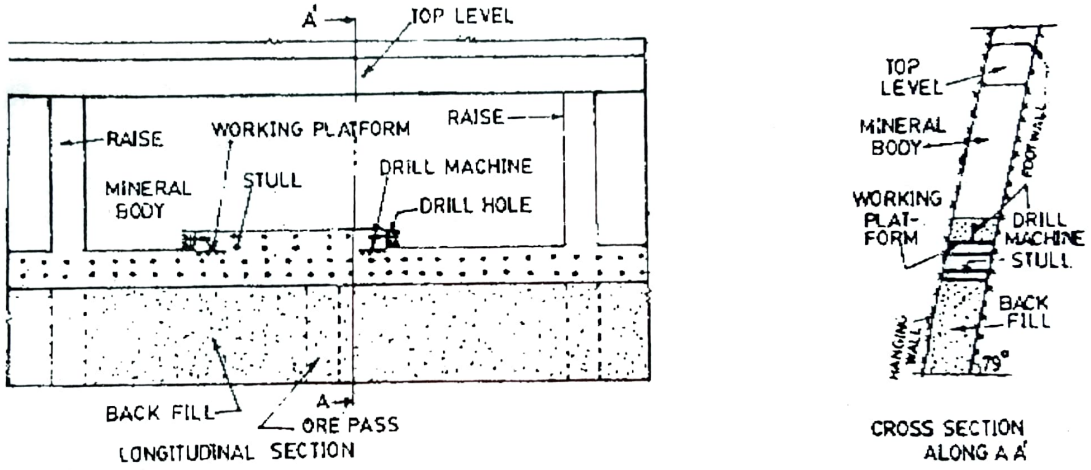


Fig. 100 Stulled Stoping

necessary, although the stope faces are supported by the stulls. The method is applicable in a thin, regular and high value mineral vein in which the hanging and foot walls are quite stronger so that stulls may provide sufficient supporting to them. Stulls are placed judiciously in the stope depending upon the site specific conditions. It is already mentioned that in this method the hanging wall should be quite strong (i.e. not very weak) so that the stoping faces can be advanced a few metre ahead without any instantaneous supporting by the stulls. If the hanging wall is very weak then stulls will be incapable to support the hanging wall hence other method of stoping should be applied in that case. It is a very costly method of stoping since high cost timbers (stulls) are being used in this case. Presently due to scarcity of trees by destroying forests and also considering the environmental safeguard from pollution, timber should be avoided to use as far as practicable in any stoping operation. Hence

except in a very small mine having a very high value mineral deposit this stull method of stoping should be avoided. This method already became obsolete since it is highly labour intensive, less productive, costly and presently very difficult to procure timbers for supporting.

(B) SQUARE SET STOPING

This is also either an underhand or overhand method of stoping in which a small block of mineral body is extracted out from the stope face and square set supporting is replaced in place of it. The square sets are connected to each other forming a continuous timbered skeleton structure made by rectangular frames which are supported by legs (posts) at their corners and over the legs (posts) again rectangular frames are placed. The square set method is applicable when the orebody and wall rocks are very weak and strong instantaneous supporting is demanded which is practised with the help of square sets, which are erected and continuously kept extended for immediate supports both in the vertical as well as in the horizontal direction in the space erected by the extraction of ore as shown in the fig. 101 (overhand method of square set stoping). This

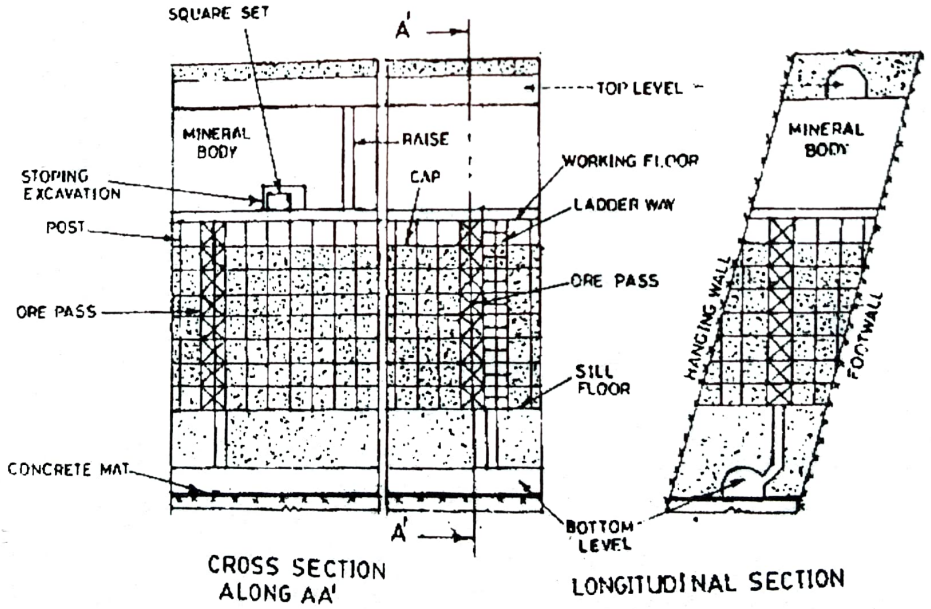


Fig. 101 Square Set Stoping

method is also applicable for stoping very geologically disturbed remnant pillars. The method requires a heavy labour force to erect square sets, cost of production is very high because of higher cost of the timber; less productive, operation process is very slow, requires skilled miners, less safety since presence of fire hazard in this case is more, etc. Presently for

the protection of surface environment and also due to the large scale destruction of forest, the availability of timber also become very difficult. In case of overhand square setting method as far as possible complete sets are erected but wherever a complete set cannot be erected on account of limited space as may happen in the neighbourhood of ore boundaries even after some nicking into the walls, a diagonal set is erected. All sets in a horizon from a floor and the bottom-most floor is known as a sill floor. As stoping progress upwards, additional floors are laid, but before starting the third floor, the sill floor is filled with filling material to strengthen the supports. This filling also progresses upwards with stoping keeping the top-most set vacant to provide working space. Some of the sets are left unfilled by enclosing them with wooden side loggings to provide passageways for ore chutes and ladderways. The square setting may be resorted right from the commencement of stoping in which case the sill floor is laid over the solid ore or an ordinary cut and fill stope may be converted into a square set stope as and when necessary in which case the sill floor is laid over the fill itself.

Each square set consists of posts, caps and struts. Posts are vertical members whereas caps and struts are horizontal members laid along the dip and strike directions respectively. The initial square set is made up of 4 posts and 2 caps and 2 struts. Further extension along the same horizon requires only 2 posts with 2 caps and one strut or 2 posts with 2 struts and one cap. These posts, caps and struts are made of round seasoned timber of about 25 to 30 cm. in diameter and are 1.98, 1.68 and 1.57 m in length, respectively. Both ends of each member are cut in this manner except the posts laid in the sill floor in which case the lower end of the posts are kept plain. As soon as an excavation of about 2.2 x 1.8 x 1.8 m has been made, a square set is erected. For this purpose, to start with 2 numbers 12.5 cm wide and 7.5 cm thick sill boards measuring about 2 m in length are laid on the floor along the strike of the orebody in a channel made in the fill so that they are partly buried in fill and are retained in their places. In case the floor comprises ore, the sill boards are fixed in place by wedges. Then 2 posts are erected with their plain ends resting on the 2 sill boards and fixed in place by wedges. A cap is then erected on top of these 2 posts and wedged in position. The next 2 posts with a cap in between are then erected in the same manner. These 2 sets each comprising of posts with one cap are then connected with each other by 2 struts placed and wedged in position, thereby completing the erection of

the first set. The post are also held together by tie-boards 7.5 x 12.5 cm in cross-section fixed on both sides at near mid-height of the square set, the other 2 sides of the set being left open for a passage way. The roof is then supported by laying planks 20 cm wide and 7.5 cm thick over the set. To advance the face, drilling and blasting of ore is done from the cover of the set. After the ore has been blasted, newly exposed roof is supported by forepoling of the timber planks before loading the ore. After loading the ore, the next square set is completed by erecting another 2 posts and laying connecting caps and struts as required. In this manner, the face is advanced towards the front as well as the sides completing one lift of about 2.2 m or so in height covering the full width along the full length of the stoping block. To take up the second lift, roof laggings of the starting set are removed and vertical blast holes are drilled and blasted in the roof creating enough space for erection of the first square set of the second floor. This square set known as a raise set is then erected on top of the lower set by placing posts of the upper set exactly over those of the lower set in the same manner as described before and further progress is made towards the front and sides with erection of further square sets over those of the lower set.

In this method, square sets are erected one over the other and therefore stoping of the whole block must be completed before the sill floor, i.e. bottom-most floor starts giving way due to decay of timber. Experience suggests that the size of the block to be stoped out with square setting right from the beginning should not exceed 30 m in length along the strike. This requires closer spacing of winzes at intervals of 30 m which also enhances the rate of filling on account of reduction in load involved in manual transport of the filling material.

With this method a stope O.M.S. around 0.7 to 1.5 tonne can be easily obtained.

The method becomes obsolete because of it's several demerits which are already mentioned. However this method can be practised in a very small mine, in a small scale production of a very high value mineral deposit and when condition of the wall rocks are very weak.

SHRINKAGE STOPING

Question 198 : Discuss elaborately **Shrinkage Stoping** method after enumerating its application, various advantages and disadvantages.

Answer : This is an overhand type stoping method where the slices/ benches/steps are blasted and left in the stopes for the purpose of supporting. After blasting the mineral body, the blasted ores are swelled to a factor around 1.4 to 1.6. As the drilling and blasting operation progress, the broken ore increases accumulation in the stope and slowly a few percentage of the broken ores are to be drawn out from the bottom level via. chute raises for making optimum rooms for drilling and blasting operation by the miners in the stope. In this way blasting operation and ore drawing operation are to be carried out simultaneously. After completing stoping of the total block the rest of the minerals are drawn out upto the cut off grade limit of dilution level of the ore. In weak rock conditions, this method is unsuitable since the broken ore is squeezed by the hanging and foot walls which trap the blasted ore and the ore can not flow downward for drawing out. If the wall rocks are very strong then dilution of ore in that case is practically nil. Hence total ore can be drawn out from the stope. The stope afterward may be left empty or filled up with the mill tailings/sand/other waste materials.

Shrinkage stoping is applicable if both the floor and hanging walls are competent, from narrow veins to thick veins (18 to 20 m), veins are sufficiently steep for free flowing of the ore (i.e. more than the angle of repose of the ore. Generally above 45^0 dip is the best), ore shall not be oxidising in nature (since ore has to remain in the stope for longer periods and oxidising of the ore will cause detrimental effects) and also have the free flowing characteristics, absence of stone intrusions of offshoots in the mineral bed, etc.

The amount of fines or clayey materials shall be as minimum as possible otherwise it will cause "hang-ups".

Shrinkage stopping can not be used in weaker strata condition because the sides of the stope would squeeze together and trap the broken ore so that it can not flow. The swell factor of the fragmented ore in the stope varies from 1.2 to 1.5 depending on the fragmentation process, nature and characteristics of ore body. Both the flat backed horizontal blast-holes or vertical (a little inclined away from the loading site of the LHDs) blastholes can be practised in this method. However the latter type is the most common since more ground can be drilled which may yield more amount of ore and also drilling and loading operations are independent.

The shrinkage stoping method is very economic, production cost of ore is very less, non-cyclic in operation, very easy to prepare a stoping block, no or very less amount of artificial supports (by timbering works) are necessary, blasted ore act as a platform for miners to work on it, faster operation is possible (sometimes handling of ore is necessary to level the blasted ore for operation in the stope), production is achievable with this method irrespective of drilling and blasting operation (because of stocking of ore in the stope), hadling of ore is not necessary in the stope except leveling work for installing drill machines. However, this method has several demerits like blockage of huge amount of blasted ore, i.e. in otherword blockage of capital, less flexible, unsuitable for sulphide ore, irregular deposit causes obstruction of free flowing of blasted ore, dilution of ore is very common (except in the case of strong hanging and foot wall), sorting out of the waste rock from the ore is not possible, since broken ore is always uneven in nature, the persons who work for drilling and blasting purposes in the stope face much of difficulties, control of grade of ore is not possible if the patches of walls are very weak it creates problems like squeezing of ore and hence drawing of ore is very difficult, sudden geological disturbances hinder the progress of stoping, large block of blasted ore requires secondary blasting otherwise it will block the ore chutes, before drawing out of the ore completely the stope is required to be filled up with the waste materials/mill tailings/sand, which will definitely pose very much problem, requirement of manpower is high, changing from shrinkage method to other method in the same mineral block is generally not possible, control of flowing of ore through the chute is very difficult, possibilities of spontaneous combustion of ore, both the hanging and foot walls shall be competent and should have angle more than 55^0; spalling of rock dilutes the ore, 60% of ore block up in the stope during normal operation etc.

Fig. 102 shows a brief about the typical shrinkage stoping method with drill holes for blasting. The thickness of vein is 6 m and dipping at 80^0. Top and bottom drives are driven in the mineral body near the footwall at a vertical interval of 40 m and the mineral block is divided with the help of side pillars of 15 m thick and the centre to centre distance of the pillar is 50 m. A 5 to 6 m thick crown pillar will be left just below the top drive when the total stoping will be completed in the

block. Above the bottom drive there is the sill pillar of 10 m thick. Chute raises are driven in the sill pillar at an interval of 15 m.

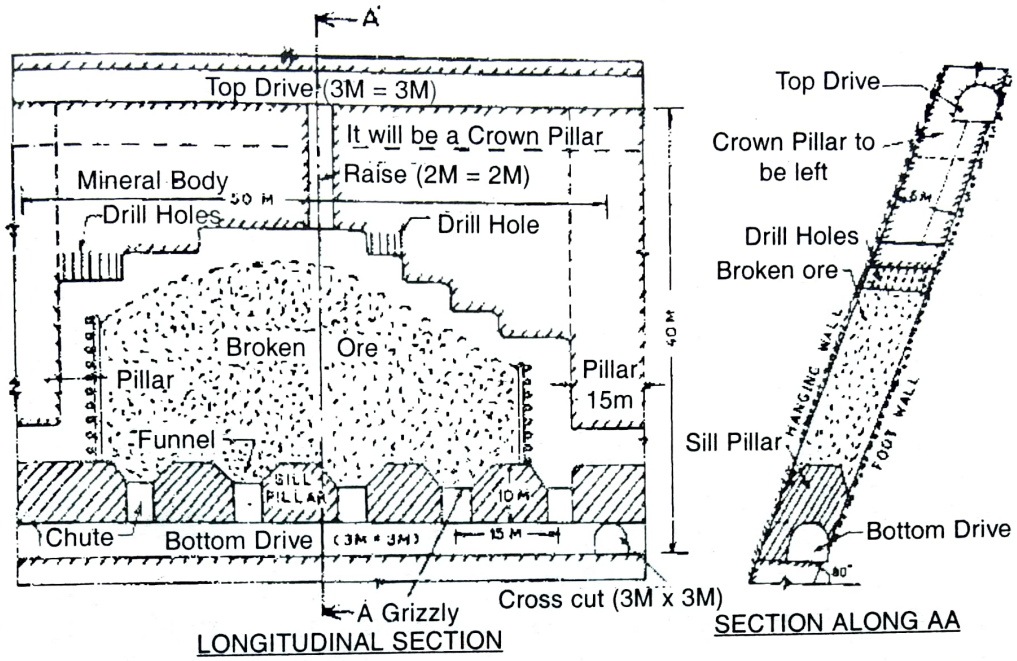


Fig. 102 Shows a Brief about the Typical Shrinkage Stoping Method with Drill holes for Blasting (not to scale)

Sometime close spacing less than 0 m is desired for free drawing characteristics. The size of the chute raises are 2 m x 1.5 m. Sizes of both the top and bottom drives are 3 m in width and 3 m in height. At the centre of the stoping block a raise of size 2 m x 2 m is driven for the purpose of ventilation, lowering of materials, man transport and also to act as a slot. Funnels are made above the top of the grizzly raise of the sill pillar. For the sake of ventilation circuit, travel of men and other purposes artificial barricades are erected (to from artificial raises) in both side of the stoping block near to the side pillars with the help of wooden stulls and planks, so that broken ore does not choke the air and men passage (here connection through the side pillars via. the artificial raises). The overhand slices/benches/steps of 2 m in thickness are flat backed and holes are drilled in the upward direction with the help of either jack hammer or stoppers. Diameter of the blastholes varies from 34 to 51 mm. In case of 34 mm dia. blastholes the burden and spacing will be around 0.4 to 0.5 m and 0.5 to 0.6 m respectively and S.G. of 60% may be used in this case with a charge per hole around 490 gms to 630 gms depending upon the hardness of mineral body. For 51 mm dia. blastholes the burden and spacing will be around 0.8 to 0.9m and 1.1 to 1.3 m respectively.

ANFO can be used with 60% S.G. as primer initiated with Nonel millisecond delay detonator and blasted with detonating cord. Drilling and blasting operation should be optimum so that too much fine does not produce otherwise these fines will cause jamming of the blasted ore. Sometimes ores are cemented together with these fines as well as if sulphide ores are stoped, since oxidation of sulphide ores will create this problem.

After blasting is over ore will be accumulated in the stope, keeping a working height around 1.8 to 2.5 m above the blasted ore, the remaining ore shall be drown out through the chutes. A stope O.M.S. of 1.2 to 3 tonne is very common.

MORE DETAILS ABOUT THE BREAST STOPING

Question 199 : Discuss **Breast Stoping** method after enumerating its application, various advantage and disadvantages.

Answer : It is an open stoping method and vertical faces of the stopes are advanced horizontally. Holes are drilled horizontally (may be inclined a little bit) in the vertical faces (called the breast of mineral body). This method is suitable when both the hanging and foot walls and also mineral body are very strong, thin to thick (say upto about 25 m), flat to gently dipping and for low grade mineral body. In this method heavy amount of ore pillars are left in the stopes and dilution of ore is very less. Since it is applicable in the flat to mild gradient mineral deposits, sorting of waste material in the stope is possible. With this method selective mining can be done very easily to maintain the grade of ore since considerable room is available for stoping. The method is very simple and requires less development work. But if the mineral deposit is very irregular in shape then development work is to be done in the unproductive country rocks. In breast stoping since pillars are left in the stopes losses of mineral is very high, hauling by scraper or other haulage system is very difficult, regular assaying is to be done to leave the low grade portion of the mineral body as a pillar. The amount of pillars will be left in the stopes depends upon the geotechnical parameters of hanging wall and mineral body, depth of the deposit, presence of geological disturbances, watery condition, etc. In the geotechnical parameters the compressive strength, tensile strength, shear strength, modulus of elasticity, poison ratio, mineralogy, lithology, thickness of the veins, dip of the deposit, etc. are

very important. Presently rock bolting and cable boltings are widely in the breast stoping. Sometimes, combination of chock supporting and rock boltings are also practised.

In the fig. 103 a typical breast stoping method with the pattern of

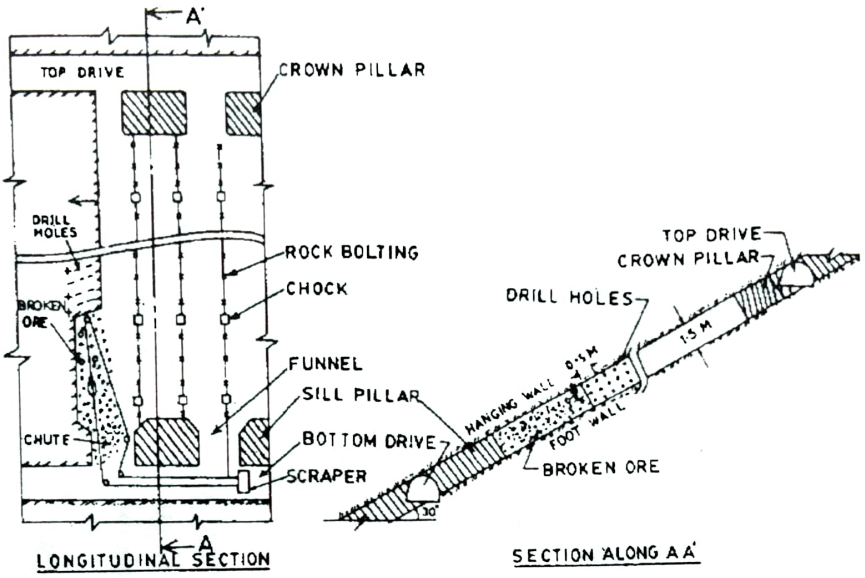


Fig. 103 Shows a Brief About the Typical Breast Stoping Method with the Pattern of Drilling and Blasting Practices (not to scale)

drilling and blasting practices has been clearly shown. The thickness and dip of the vein is 1.5 m and 30^0 respectively. Top and bottom drive has been driven in the mineral body at a vertical interval of 37m. length of the stope face varies from 66 to 70 m. Crown pillar of thickness around 8 m is left below the top drive whereas sill pillar of thickness around 12 m is left above the bottom drive. Chute raises are driven in the sill pillar at an interval of 15 m centre to centre. The details about the drilling and blasting practices, manpower, etc. are given in the question no. 183. After blasting is over ore is scraped down with the help of scraper in thc ore chute. In the stope hanging wall is supported by rock bolting and chock combination as shown in the fig. 103. A stope OMS around 3 to 5 tonne can easily be obtained with this method.

ROOM AND PILLAR STOPING

Question 200 : With the help of a neat sketch discuss **Room and Pillar** Stoping.

Answer : In this system first of all mineral body is divided by rectangular

blocks by driving top and bottom levels and connecting the level by raises thereafter. In this way room (raise) and pillars in between the raises are formed. The rooms are then extended in either side of the raises till a rib pillar around 4 to 5 m thickness is left in between the two rooms to support the hanging wall. The pillars thereafter may be robbed out as a final extraction. This room and pillar method of stoping is generally practised in the flat or gently dipping strong, cheap mineral deposit. The deposit generally should be in huge amount with more or less uniform geotechnical parameters. Both hanging wall and footwall shall be strong enough for doing open stoping.

In the fig 104 a details about the room and pillar system of mining

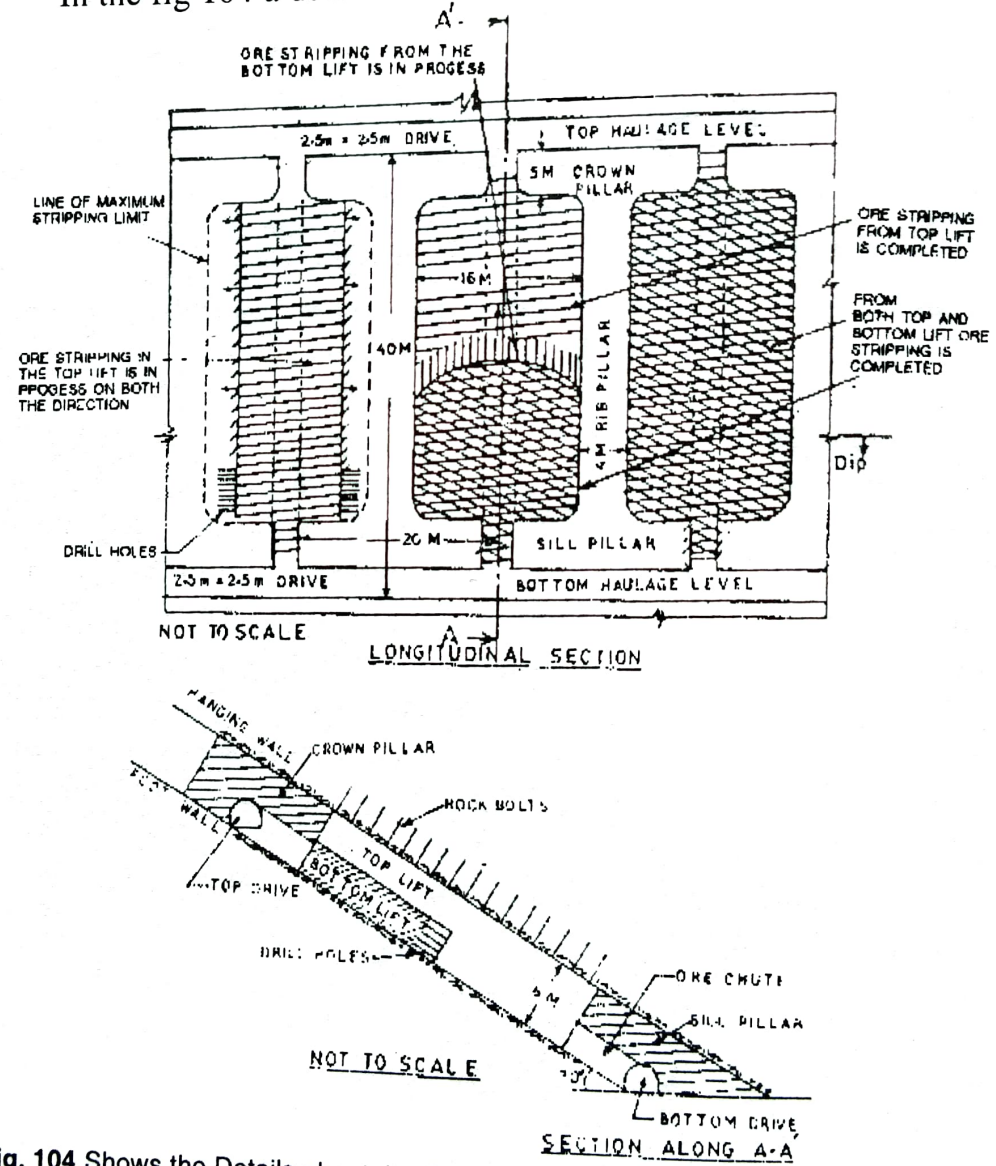


Fig. 104 Shows the Details about the Room and Pillar System of Mining along with Drilling and Blasting Pattern

along with drilling and blasting operation has been shown. The thickness and dip of the mineral body is 6 m and 30 (from the horizontal) respectively. Top and bottom levels (sizes each 2.5 m x 2.5 m) are driven in the ore body along the footwall at a vertical distance interval of 40 m. The levels are connected by the raises (sizes 2 m x 2 m) at an interval of 20 m along the strike length. Sill pillar is left at the top of the bottom level with a vertical thickness of 5 m. Same way crown pillars are left at the bottom of the top level with a vertical thickness around 5 m. Bottom of each raises acts as an ore chute. The 6 m thick mineral body is extracted in 2 lifts of 3 m each. First of all top lift is heightening up and extracted afterward expanding the same in both the direction along the strike length as shown in the fig.104 for enlarging the rooms. A 4 m width rib pillar is left in between the two rooms for supporting the hanging wall. The hanging wall is supported by grouted type rock bolts at a bolts density around 1 bolt per 1.44 m^2. The length and diameter of the rock bolts are 1.5 m and 22 mm respectively. After taking the top slice (room of 16 m in width), the 3 m bottom slice is stoped from dip to rise direction i.e. from top of the sill pillar upto the bottom of the crown pillar. In this way the stoping of full height of 16 m room is completed. The 4 m rib pillars may be robbed out juduciously later on.

The blastholes of 1.5 m length and 34 mm diameter are drilled in the horizontal direction with the help of leg mounted jack hammer at a spacing and burden of 0.5 m and 0.5m respectively. Explosive used is 60% S.G. with a charge per hole around 350 gm to 490 gm (approx.) depending upon the site specific condition of the mineral body. After blasting is over the blasted ore can be scraped by the scraper to discharge into the ore chute. From the ore chote ore is loaded onto the mine cars of capacity 2.5 tonne and hauled to the required destination by the mine locomotives/haulages. A stope OMS around 2 to 5 tonne is quite common with this method if scraper is utilized to load the blasted ore into the ore chutes.

CUT AND FILL STOPING

Question 201 : Discuss elaborately Cut and Fill stoping method after enumerating its application, various merits and demerits.

Answer :

In this stoping method ore is extracted out by overhand method of

stoping forming flat/or inclined slices. After removal of the blasted ore the space is filled with either mill tailings or sand or waste material, etc. leaving a working space about 2 to 2.5 m above the filled material for enabling the miners to work over it for taking the next slice without any difficulties. This method is applicable in weak to strong hanging and footwall, from moderately thick to quite thick deposit (20 m or more) at any condition, any gradient and any depth range, when ore is quite strong, (so that the back of the slices are kept intact still blasting is practised), where grade of ore is quite rich and highly valuable, where there is the chances of fire hazards, etc. The merits of this method includes higher safety and recovery, lower dilution of ore and development work, nicely filled material strongly supports the weak walls and also reduces surface subsidence, ventilation is easy, sorting of waste material from the ore is possible in the stope, the problem of surface disposal of mill tailing can be eliminated very much if the mill tailings are used as filling material, stope supervision is very easy due to cyclic and concentration of work, manpower requirement is low, possibilities of quicker and simultaneous extraction of various levels at a time at various faces, quicker return of capital, etc. But the method has several demerits, like higher production cost of ore, consistent procurement of filling materials are very difficult and costly, for consistency of production several stoping blocks are to be operated simultaneously, if inclined slicing cut and fill method stoping is practised then several problems will be faced like drilling, extraction of top portion of the block, delay due to making platforms for workers, etc.

Excessive ore shall not be drawn through the chute otherwise wooden flatform for drilling and charging have to be made and there may be the chances that cavity may form in the hanging wall due to the caving of the strata. During stoping operation the hanging wall may be supported by the roof bolting or by the natural mineral pillars. The natural mineral pillars shall be either round shaped or be tapered at the top to a slope of about 55^0 to 65^0 to avoid ore hang-ups over them. After finishing the stoping operation all the broken ore shall be taken out keeping the stope void. Thereafter it may be tightly back filled with the filling materials (mill tailings, sand etc. and may be mixed with cement) or total void area shall be filled up with the caved goaf (it is done by blasting off the Crown pillars and remnant pillars which are left in the

goaf during the normal operation).

However the cut and fill stopping has the following demerits too :

- filling material must be readily available for this system as and when necessary. Availability of filling materials and their cost including the cost of transportation and filling may account 40% of the total mining cost.
- problem on disposal of tailing slimes in a systematic manner.
- difficulties in manoeuvrability of LHD, Drill jumbo, etc. machinery in the stopes. Servicing and maintenance etc. shall be carried out in the stope itself. Under adverse condition and major breakdown the machinery shall be dismantled and lifted out from the stope to the local workshop.
- machinery can not be shifted easily from place to place to deal with the fluctuation of production demand.
- the blast-holes shall be drilled at an angle of 70^0 to 80^0 from horizontal leaning away from the loading direction to avoid any back break and also for efficient loading of ore by the LHDs. This is a precise and cumbersome job.
- there are acute chances of dilution of ore with the fill materials if proper care is not taken. To extract crown pillar or to stabilize the fill around the ore pass or ventilation raise a 7% to 10% cement shall be mixed with the fill materials.
- this method is suitable for low to medium rate of production.

Cut and fill stoping may be conventional or mechanised. In the fig.

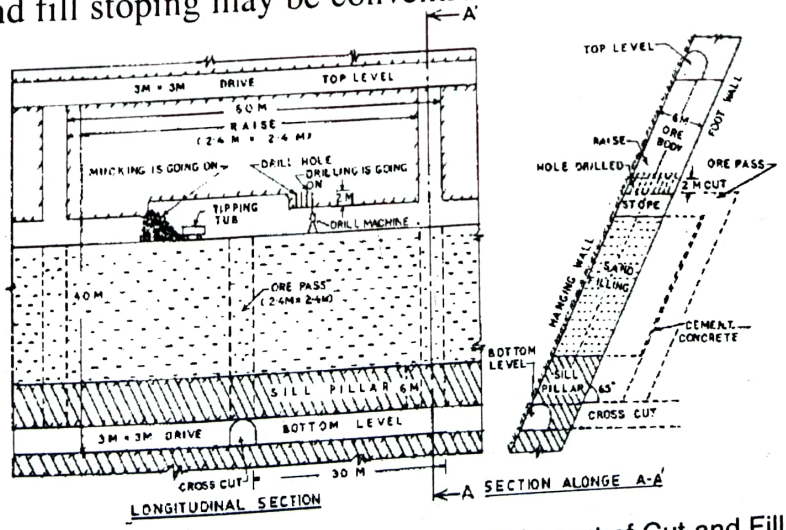


Fig. 105 Shows the Details about the Conventional Layout of Cut and Fill Stope Showing the Drilling Pattern for Blasting (not to scale)

105 the details about the conventional layout of cut and fill stoping along with the drilling pattern for blasting has been shown. The deposit is 6 m thick dipping at an angle of 65^0 from the horizontal. Top and bottom levels (both are 3 m in width and 3 m in height) are driven at a vertical interval of 40 m along the hanging wall. The levels are connected by the raises at an interval of 60 m along the strike (centre to centre distance) and hence completed a stoping block. For ore pass first of all a cross cut is driven from the bottom level to the country rock and a raise is allowed to from simultaneously as the stoping face is progressed upward (like shrinkage stoping). For protecting the ore pass form the sand filling material cement concrete slabs are placed in the mouth of the vertical portion of the artificially made ore pass. Except a sill pillar of height 6 m no other pillars are left in the stope. The slices are flat backed which is drilled with the help of conventional type of drill jumbo or leg mounted jack hammer. The holes are vertical and height is around 2 m. The diameter of the drill holes are 34 mm, burden and spacing varies from 0.4 to 0.5 m and 0.5 m to 0.6 m respectively. Explosives used is 60% S.G. with a charge per hole around 490 gms to 630 gms depending upon the hardness of mineral body. The blasted ore is shovelled manually over the tipping tub which is transported and unloaded ore into the ore pass.

From the bottom of the ore pass ore is collected over the side tipping tubs or in ordinary mine tubs/mine car which are afterward hauled either by the underground mine locomotives or rope haulages. The total 60 m length (along the strike direction) of the block can be divided into 2 parts of 30 m each in either side of the ore pass. When in one side mucking will be carried out (after blasting), in the other side drilling will be done. When mucking will be over the drill machine will be shifted in the previous mucking side while in the other side blasting of the drilled holes and afterward mucking will be carried out. After completion of one slice of 2 m height, leaving 2 to 3 m, the remaining portion will be backfilled with the rejects of the stopes or mill tailing or sand or crushed overburden of the opencast mine. In this particular case sand is filled into the stope. If required temporary supporting with the help of stulls of chockmats can he done over the filled packing. Since loading is done here manually sorting of waste material can be done manually in the stope. In this type of high manpower intensive stoping a stoping OMS around 1.5 to 3 tonne can he achieved very easily.

In the fig. 90 a most modern cut and fill method has been shown which has been discussed in the question no. 184. Here Electro-Hydraulic drill Jumbo is used for drilling and 2.3 m^3 bucket capacity LHD is used for cleaning the broken ore. With this method a stope OMS around 35 to 40 tonne is quite common.

The system eliminates completely less flexible, less productive rail mounted haulage equipments with the tyre mounted machines, method is highly productive, dilution of ore can be reduced, selective stoping is possible, highly flexible, can maintain the required grade of ore very efficiently, extraction of high percentage of ore is possible, reduce the number of manpower heavily in the stope, the tyre mounted LHD can load and transport ore around 250 tonne/hr. If necessary either rock bolting or cable bolting can be introduced in the hanging wall. First of all with the help of 150 to 200 mm pipe range sand, along with water mixture in the ratio sand : water (by volume) = 1:1.9 and height: length ratio = 1:8 is taken in the underground above the stope from where sand and water mixture is thrown into the stope with the help of 100 to 150 mm pipe range. Temporary wooden barricades are erected judiciously wherever necessary during back filling. The rubber tyred vehicle in the mechanised cut and fill stoping transport ore right from the stope to the required ore passes. The inclined access into the stope as shwon in the fig.105 facilitates maximum utilization of any material handling equipment, over-hauling of equipments in the underground itself, and direct transport of ore right from the stope to the required dumping place.

SUB LEVEL STOPING

Question 202 : Discuss elaborately **Sub-Level Stoping** method after enumerating its application, various merits and demerits.

Answer :

When a small and irregular ore body encapsulated by weak rock, the block caving method will cause excessive dilution. If the ore body is hard then block caving method can not operate efficiently since free flowing characteristics of ore is missing. Under such condition sub-level caving may be practised in place of block caving. For selective mining of variable graded ore this method is better. For grade control and maintaining excellent flow properties the sub-level interval shall be within 12 m. It will also assist of restricting longer blasthole length in the

ring drilling. Drilling of longer blasthole are very time consuming, combersome and very difficult to control the hole deviation. The fans are generally drilled at an angle of 75^0 to 80^0 from the horizontal leaning towards the goaf to reduce the back break and sliding friction.

Recovery rate in the controlled manner from the stope and grade control is the must in sub-level caving system since dilution rises rapidly at the end of a run. The ore drawn must be precalculated and the quality drawn must be carefully sampled either in the conventional method ore by the rapid analysis instrument. A compromise between dilution and recovery rate must be formulated.

Is is an open stoping method. In this method the main haulage levels are driven at an interval of 60 to 100 m and sub levels are driven in between the haulage levels for making ring drilling and blasting operation from bottom to top sub levels. The method is suitable for a regular mineral body having an uniform grade and a dip of more than 50^0 for easy free flowing of the blasted ore under gravity. The mineral body shall be moderately strong whereas the hanging and footwalls should be very strong for minimising the dilution of ore, etc. In mechanised sub level stoping the ore body should be sufficiently wide to accommodate the diesel powered drill jumbo, LHD etc., machinery, the site shall be located in a remote area where higher production and productivity will be required, etc. The method is highly productive, economic (cost per tonne of ore is less), less labour intensive (provided mechanisation is done), can be used for sulphide ores, well ventilated because of sublevels, and a very safe method. It allows selective mining of weaker or medium strength ore bodies, flexible method, mechanisation can be done efficiently, development is less compare to block caving, no pillar of ore is left in the goaf, can be used in wet and sticky ore, etc. But sorting of waste material from ore in the stope is not possible, the method is not suitable in the geologically disturbed area, causes surface subsidence, dilution level is high (around 25%) etc. Since longhole ring drilling is one of the very important features, efficient jumbo drills (Simba-25) should be employed in the stope. Persently in sub-level stoping all operations are carried out from the same level. The ring drilling fans are all parallel and the cones are automatically formed in the blasted ore. In narrow veins the haulage drift is situated in the country rock. Mucking of ore can be done with the help of rail mounted rocker shovel or with trackless LHD or CAVO-320 or CAVO-520 loaders.

In the fig. 106 a brief about a typical sub-level stoping method with

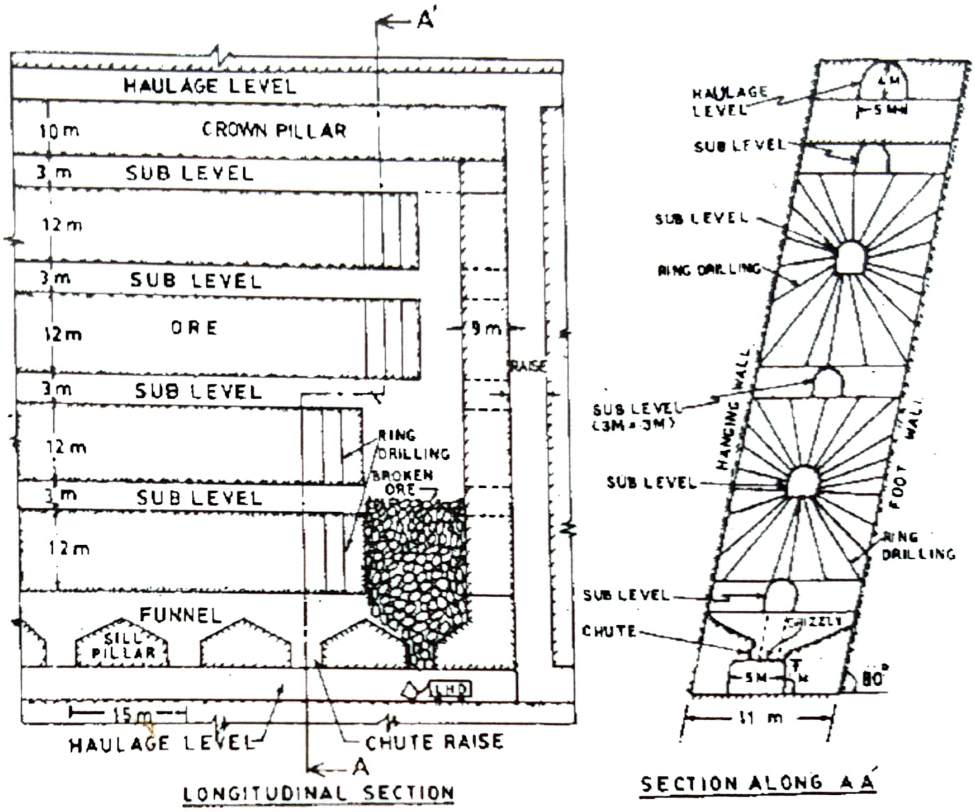


Fig. 106 Shows a Brief about the Typical Sub-level Stoping Method with Ring Drilling and Blasting (not to scale)

ring drilling and blasting has been shown. The thickness of ore body and the dip is around 11 m and 80^0 (from horizontal) respectively. Both the bottom and top levels (sizes 5 m width x 4 m height) are driven through the centre of the mineral body. The vertical distance between the two levels is kept to 82 m. In the block below the haulage level crown pillar (10 m thick) and in the side after level connecting raise a 9 m side barrier pillar is left in between the two stoping blocks. Above the bottom haulage level sill pillars of height around 8 m is left. Chute raises are driven at an interval of 15 m (centre to centre) through the centre of the ore body. After development is over ring drilling is done in the vertical planes radially like a fan with the help of Simba-25 drill machine as shown in the fig 106. The diameter of the drilled holes are 51 mm and the length varies according to the angle of the holes in the ring. Main explosive used here is ANFO with 60% S.G. as primer, initiated with short 25 milli-second delay detonator between holes in each row of ring starting from the easiest breaking section in the middle and progressing

towards the walls. Longhole drilling is very costly but it is overall economical since it saves the meterage drilled, cost of blasting accessories, and the initial development cost. Explosive consumption is around 0.15 to 0.3 kg per tone of ore blasted. During blasting operation first of all one bottom ring is blasted and blasted ore is collected by the LHD in the haulage level. After drawing sufficient amount of ore, the next bottom ring is blasted and ore is drawn out through the chute. In this way a number of rings are blasted in the bottom ring stoping face. When this face will be advanced quite sufficiently blasting operation in the top ring stoping face shall be commenced. A time will come when simultaneous blasting of both the top and bottom rings can be carried out. Bucket capacity of each LHD is 2.3 m^3 which can easily load and transport ore around 250 tonne/hour. In this system a stope OMS around 15 to 25 tonne is quite common.

Question 203 : With the help of an example discuss sub-level stopping with mass firing

Answer :

Example

Sub-level Stopping with Mass Firing

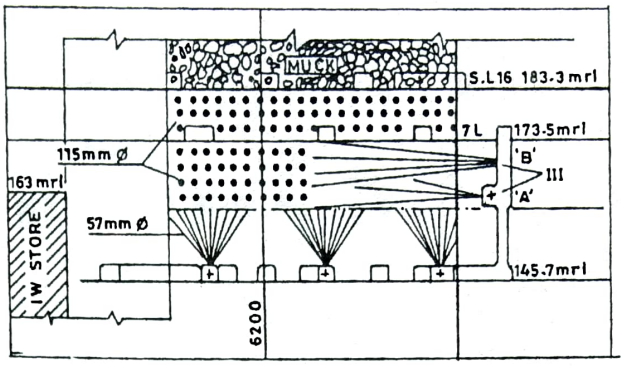
Size of stope : 60 m long, 39 m thick, 30 m wide

The lower 12 m portion of the stope has been excavated as trough and draw point at the bottom and the rest portions are the stope. Thus remaining 27 m block is divided into 3 slices of 9 m each. Each slice was drilled with 115 mm dia. DTH holes in the form of horizontal/sub-horizintal fan rings drilled at the burden and spacing of 3m x 3m from offseam drilling chambers. Drilling of these slices were done by two sets of drilling chambers - one situated on eastern extremity of the stope having chambers A and B and the second form drilling chambers situated in the footwall of the ore body. They are the C, D and E chambers. These chambers are located at about 10 m from the stopping block for safety purposes.

The 27 m thick stoping block was fired horizontally from top to bottom in the trough below by sequential blasting of the slice drilled from chambers A and C, followed by slice covered by chamber B and D and that covered by the chamber E at the last.

Details of the blasting practices (Fig. 107)

LONGITUDINAL SECTION



TRANSVERSE SECTION

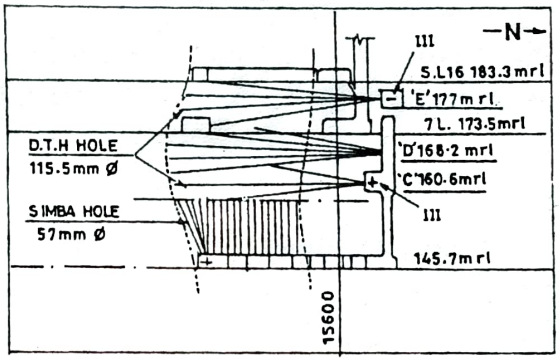


Fig. 107 Sub-level stopping with mass firing

Hole diameter (mm) : 115
Burden (m) : 3
Spacing : 3
Total number of holes
in the mass blast : 357
Explosives used : in dry hole - ANFO along with cast booster PRIMEX
in wet hole - OCG along with cast booster PRIMEX

The maximum charge per delay (Kg) - 1860
Total explosives used (tonne) - 43
Detonator used : non-electric milli-second delay detonators. (PRIMEX is initiated by ANOLINE), S.G. is

initiated by ANODET. ANOLINE is attached with the trunk line of 10 gm CORDTEX detonating cord.

Delay between the rows of holes (ms) : 60

Delay between the blast-holes (ms) : 25

Powder factor achieved (tonne/Kg) : 3.1

The detonators were fired by a conswigear 200 shot (capacitor-discharge type) exploder.

Time taken to fire the whole of the stope : 3.375 seconds

Prior to stope blast, the draw-point pillars and troughs were constructed by drilling and blasting. The periphery of the draw point pillars should be suitably pre-supported by means of rock-bolting and wiremesh, etc.

Stope under the envelope of high stress condition could be mass-blasted without causing damage to the main mine structures by restricting charge per delay.

Drills Used :

1) 4 BBC 120 F drifter mounted with Simba Junior Rigs (57 mm diameter blast-hole)
2) 2 COP - 4, down - the-hole hammers (115 mm diameter blast-holes)

Production : 130000 tonne by a single mass blast.

BLOCK CAVING

Question 204 : Discuss elaborately Block Caving method after enumerating its suitability, various advantages and disadvantages.

Answer :

In this method a huge block of mineral body of height around 18 m to 100 m or more is caved down by giving an undercut. The block is caved down and fragmented by its own weight if the mineral body is very weak, otherwise the mineral block is caved down by drilling and blasting method. The method is suitable for a very wide veins, weak hanging and foot walls and also mineral body which after giving an undercut will cave down very easily and fragmented. The block caving method (without induced blasting) is possible only when the mineral body will cave down

and well fragmented just after giving the under cut i.e. mineral body is sufficiently weak, whereas hanging and footwalls are stronger, thickness of the low grade mineral body is around 18 to 32 m, etc. In the case of block caving method (with induced blasting) the strength of ore body may be a little bit stronger compare to the former case. But in this case more amount of development works are needed for drilling and blasting operation. Block caving method offers a very high degree of production, safety and productivity, attains a very low production cost, requires very little amount of supporting materials and explosives, provides consistency of production from the block after caving is over, etc. But the method has various demerits too like huge amount of ore losses in the block, high rate of dilution of ore by the country rock, control of ore drawing operation is very difficult, selective mining as well as sorting of ore in the stope is not possible, blockage of invested capital for a longer period, necessitates a high degree of preliminary exploration, etc. Due to the extensive caving operation this method always associated with surface subsidence.

In the fig. 108 a typical layout of a block caving method with

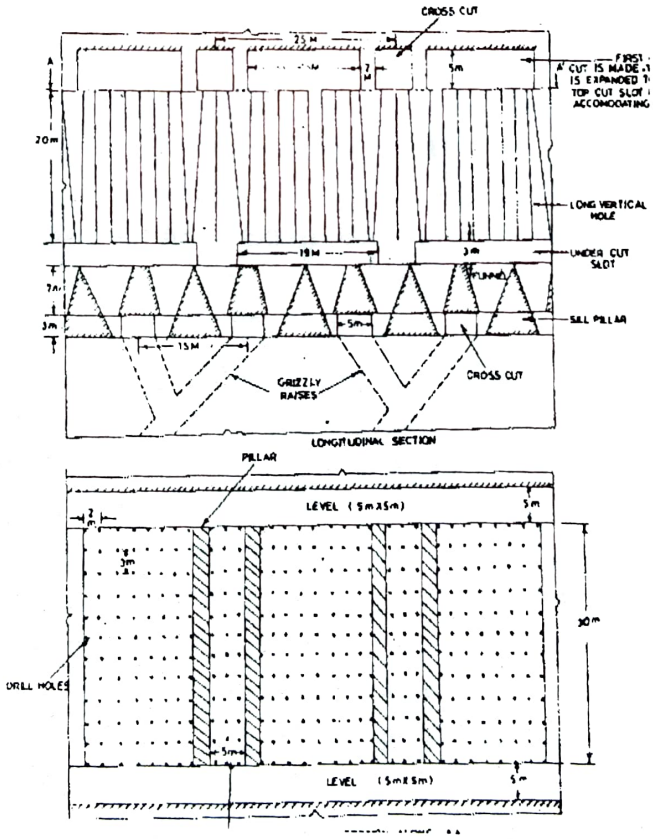


Fig. 108 Shows a Typical Layout of a Block Caving Method with Long Hole Drilling

longhole drilling has been shown, The width of the mineral body is around 40 m, and dipping at 90^0. The thickness of the mineral body is around 35 m. At the roof of the mineral body levels, (5m x 5m) both in the hanging wall as well as in the footwall, are driven which are then connected by the cross cuts (size 5 m x 5 m). Mineral blocks are formed along the strike length at an interval of 25 m (centre to centre in between the two cross cuts). In between the two cross cuts another cross cut of size 5 m x 5 m is driven which is afterwards widened to 15 m leaving a barrier pillar of 2 m thick in both the sides of the slot. The top slot as well as cross cuts (5 m width) are maintained at 5 m height to accommodate heavy drill machines like crawler mounted ROC-306 drill machine with booster compressor of model DG-418 pack by Atlas Copco and also load haul dumpers. At the bottom of the ore block level drifts (5 m width x 3 m height) are driven in both the footwall and hanging wall of the mineral body which are connected by the cross cuts at an interval of 15 m centre to centre. Sill pillars of height 10 m are left over the floor of the deposit. Chute raises are drawn above the whole of the cross cuts at an interval of 5 m which are made funnel shape to accommodate the blasted ore and for free flowing of the same. Hence every funnel throughout the area of the mineral body may be considered as drawing excavation points. Above the drawing excavation points under cut slots are made of height 3 m, width 19 m and connecting the foot and hanging walls. To tackle the premature caving of the block of mineral body above the under cut slots pillars of 5 m width in between the two undercut slots are left. From the bottom of each funnel grizzly raises are driven in the country rock which are finally connected to the main haulage level. Holes of sizes 165 mm diameter are drilled by the ROC-306 drill machine from the top slots as well as from the cross cuts the undercut slots as shown in the fig. 108. Holes drilled from the top slots are most of the vertical however holes from the cross cuts are drilled both the vertical as well as inclined holes (around 85^0) in the downward direction. Burden and spacing of the holes are 2 m and 3 m respectively. If any drilled hole punctures the under cut slot then first of all the bottom of that hole is plugged with the help of a nylon cord and afterward stemming is done upto a height of 1 m above the plug. A drilling rate of 30 m to 40 m per shift is possible with the machine provided the supplied pressure by the portable booster compressor is very good. ANFO can be used here to blast the mineral block. First of all ANFO shall be poured around a

height of 1 m above the stemming and Pentolite cast booster alongwith Raydet which will act as primer shall be placed over it. To reduce the ground vibration holes may be charged with decks which will also reduce the amount of explosive consumption. Therefore, if the deck charging is practised then from the Pentolite cast booster ANFO will be poured upto the predetermined deck height. In between the two decks a 3 m space shall be filled up with sand stemming and also at the end (collar of the hole) stemming shall be done to a height around 2 to 3 m. Amount of ANFO in each hole is around 290 to 310 kg (approx).

ANFO is loaded with the help of pneumatic pressure by a pressurised ANFO loader filled with a charge up truck. Capacity of the ANFO loader is around 10 tonne. Presently the deck charging system is eliminated. Two cast booster may be used in this case (double priming system) since the diameter of thc hole is 165 mm. First of all a part of the block is blasted and the blasted ore flows freely from the stope to the draw excavation points and finally upto the main haulage level via grizzly raises. When enough free face is obtained then all the holes are blasted subsequently to cave down the total mineral block with good fragmentation of ore over the draw excavation points. The ore is collected thereafter from the chutes (built in the grizzly/raises) in the main haulage level consistently. A face OMS and district OMS of 40 to 50 tonne and 10 to 12 tonne respectively are very common in this method.

STOPING WITH LONG HOLE DRILLING AND BLASTING

Question 205 : Discuss briefly about a most modern Stoping method with Long Hole Drilling and Blasting.

Answer : In the fig. 109 a most modern open stoping method with longhole drilling and blasting practices has been shown. The thickness of the deposit is 20 m dipping at an inclination of 80^0 from the horizontal. Two levels (one in the footwall side and the other one in the hanging wall side) of sizes 4 m height, 5 m width are driven along the strike direction through the mineral body. Below these levels at an vertical interval of 40 m another two levels of the same sizes and the same fashion are driven. Both the lower and upper levels are connected by the cross cuts (sizcs 4 m height x 5 m width) at an interval of 30 m. Raises (2.4 m dia) are driven to connect both the upper and bottom levels of both in the

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footwall and hanging wall sides. These raises are drilled by the raise boarer machines. Afterward a mineral block excavation at the upper level cross cuts are done both in the country rock as well as in the mineral body (sizes around 5 m width x 22 m thick across the mineral body x 4 m height) for accommodating drill machine for making slot for creating a free face for ring drilling and blasting in the stope. For making slots, 165 mm dia. holes are drilled with the help of ROC-306 drill machine along with a booster compressor of model DG-418 pack by Atlas Copco. The spacing and burden of holes are 3 m and 1.5 m respectively. Rate of drilling is around 30 to 40 m per shift. After drilling is over for making

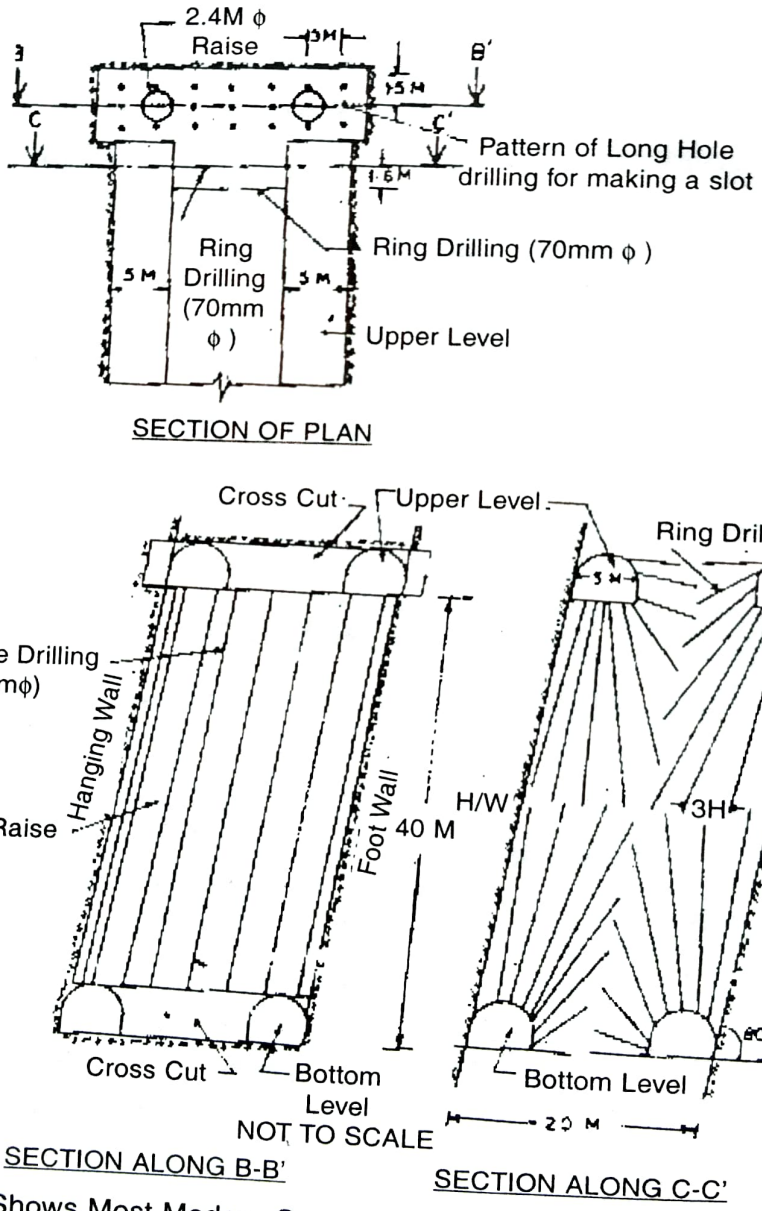

Fig. 109 Shows Most Modern Stoping with Longhole Drilling and Blasting

slot, first of all the first row of the holes are blasted with the help of ANFO and Pentolite cast booster along with Raydet which acts as primer are placed over it. If any drill hole punctures the bottom level and cross cut then first of all the bottom of that hole is plugged with the help of a nylon cord and afterwards stemming is done upto a height of 1 m above the plug. ANFO shall be poured around a height of 1 m above the stemming and Pentolite cast booster (2 in number at a place) along with Raydet are placed over it.. Amount of ANFO used in each hole is around 650 kg (approx). Another cast booster is used in the hole at a distance around 15 m from the double priming cast booster. In this way blasting of first, second and third rows are completed successively for making the slot. Ring drilling and blasting is practised in the stope as shown in the fig. 109. Diameter of the ring drilling blastholes are 70 mm, distance between the two rings are 1.8 m and the toe spacing of the ring drilling holes are maximum around 3 m. At a time only one ring shall he blasted. Amount of explosive requirement in the ring holes around 0.4 to 0.45 kg per tonne of ore produced. Pressure loader is used to load ANFO into the blast holes and drill cuttings are used as stemming material. With this method a stope OMS around 20 to 30 tonne can be easily achieveable; After completion of this block it shall be filled up with back filling materials like sand or mill tailings or well crushed overburden rocks of opencast mines, etc. This method offers higher production, productivity at lower production cost and also exposure of hanging and footwalls can be controlled nicely. To control dilution of ore controlled blasting technique, rock or cable bolting in the stopes, etc. shall be practiced.

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About the Author

Dr Samir Kumar Das (born on the 15th October 1956) obtained B.Tech (Hons) in Mining Engineering from Indian School of Mines, Dhanbad, in the year 1979. He also obtained First Class Mines Managers Certificate of Competency of the Government of India and M.Tech in Mining Engineering with distinction from Indian School of Mines, Dhanbad in the year 1983 and 1984 respectively. He obtained his Ph.D degree in Longwall Mine Mechanisation from Indian School of Mines, Dhanbad in the year 1995 and also obtained PGDHRM degree from IGNOU in the year 1997. He joined Coal India Ltd. in 1979 as a Junior Executive Trainee and then became Assistant Colliery Manager and served for 5 years in the Moonidih Project of BCCL, India. In September 1984 he joined Indian School of Mines, Dhanbad as an Assistant Professor and served there for more than 12 years. He is the founder member of Longwall Mine Mechanisation of Indian School of Mines, Dhanbad. He joined Indian Institute of Technology, Kharagpur in the year 1997 as an Associate Professor in the department of Mining Engineering. He is the recipient of many National and International awards for his original contributions in the area of rock mechanics and ground control and his research works are well recognized Internationally. Dr Das to his credit about 85 published research and technical papers (in various International and National Journals and Conferences) and many more are on the pipe line of publication. He has written 8 technical text books out of which 4 books viz. Modern Coal Mining Technology, Surface Mining Technology, Explosives and Blasting Practices in Mines, Mine Management Legislation and General Safety (coal) are already published.

In the literary world he has already established a distinguished marks with the publication to his credit three books on his collection of classic romantic poems in Bengali named 'Jhoro Batas', 'Tumi Amari' and 'Maha Sunner Jhankar' and many more may be published in the coming future. His area of interest in the literary world is the classic romantic poetry.

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